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PLACER MINING
SURFACE ARRANGEMENTS AT ORE MINES
PRELIMINARY OPERATIONS
ORE MINING
SUPPORTING EXCAVATIONS
ASSAYING

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PREFACE

The International Library of Technology is the outgrowth of a large and increasing demand that has arisen for the Reference Libraries of the International Correspondence Schools on the part of those who are not students of the Schools. As the volumes composing this Library are all printed from the same plates used in printing the Reference Libraries above mentioned, a few words are necessary regarding the scope and purpose of the instruction imparted to the students of—and the class of students taught by—these Schools, in order to afford a clear understanding of their salient and unique features.

The only requirement for admission to any of the courses offered by the International Correspondence Schools is that the applicant shall be able to read the English language and to write it sufficiently well to make his written answers to the questions asked him intelligible. Each course is complete in itself, and no textbooks are required other than those prepared by the Schools for the particular course selected. The students themselves are from every class, trade, and profession and from every country; they are, almost without exception, busily engaged in some vocation, and can spare but little time for study, and that usually outside of their regular working hours. The information desired is such as can be immediately applied in practice, so that the student may be enabled to exchange his present vocation for a more congenial one or to rise to a higher level in the one he now pursues. Furthermore, he

wishes to obtain a good working knowledge of the subjects treated in the shortest time and in the most direct manner possible.

In meeting these requirements, we have produced a set of books that in many respects, and particularly in the general plan followed, are absolutely unique. In the majority of subjects treated the knowledge of mathematics required is limited to the simplest principles of arithmetic and mensuration, and in no case is any greater knowledge of mathematics needed than the simplest elementary principles of algebra, geometry, and trigonometry, with a thorough, practical acquaintance with the use of the logarithmic table. To effect this result, derivations of rules and formulas are omitted, but thorough and complete instructions are given regarding how, when, and under what circumstances any particular rule, formula, or process should be applied; and whenever possible one or more examples, such as would be likely to arise in actual practice —together with their solutions—are given to illustrate and explain its application.

In preparing these textbooks, it has been our constant endeavor to view the matter from the student's standpoint, and to try and anticipate everything that would cause him trouble. The utmost pains have been taken to avoid and correct any and all ambiguous expressions—both those due to faulty rhetoric and those due to insufficiency of statement or explanation. As the best way to make a statement, explanation, or description clear is to give a picture or a diagram in connection with it, illustrations have been used almost without limit. The illustrations have in all cases been adapted to the requirements of the text, and projections and sections or outline, partially shaded, or full-shaded perspectives have been used, according to which will best produce the desired results. Half-tones have been used rather sparingly, except in those cases where the general effect is desired rather than the actual details.

It is obvious that books prepared along the lines mentioned must not only be clear and concise beyond anything

heretofore attempted, but they must also possess unequaled value for reference purposes. They not only give the maximum of information in a minimum space, but this information is so ingeniously arranged and correlated, and the indexes are so full and complete, that it can at once be made available to the reader. The numerous examples and explanatory remarks, together with the absence of long demonstrations and abstruse mathematical calculations, are of great assistance in helping one to select the proper formula, method, or process and in teaching him how and when it should be used.

This volume contains papers on the subjects of placer mining, surface arrangements at ore mines, preliminary operations at ore mines, ore mining, supporting excavations, and assaying. The papers are worded to furnish practical information in a concise way, and therefore will be found helpful to placer miners, mining engineers, mine managers, mine foremen, students, and others interested in mining. In some respects the subjects are presented differently than in other books, in that the practical details adopted in the latest and most systematic methods of economic mining and assay practice are explained, where usually they are left to the student or others to ascertain by practice. Since mining practice must be varied to meet conditions, this feature alone should appeal to every individual at all interested in mines.

The method of numbering the pages, cuts, articles, etc. is such that each subject or part, when the subject is divided into two or more parts, is complete in itself; hence, in order to make the index intelligible, it was necessary to give each subject or part a number. This number is placed at the top of each page, on the headline, opposite the page number; and to distinguish it from the page number it is preceded by the printer's section mark (§). Consequently, a reference such as § 37, page 26, will be readily found by looking along the inside edges of the headlines until § 37 is found, and then through § 37 until page 26 is found.

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PLACER MINING

(PART 1)

INTRODUCTION

FORMS OF DEPOSITS

1. Shallow, or Modern, Placers.—Placers are gold-bearing deposits that have been formed by the disintegration of rocks; the heavier portions of the deposits have become concentrated by the action of water. The placer deposits that are most accessible are those occurring in the beds of modern or recent rivers and streams. These have been worked from the dawn of history, and the greater part of the gold that has been obtained has come from such deposits. The material of the deposit may consist of sand, gravel, loam, or clay.

Shallow deposits may occur in the beds of rivers, or as bars along river banks or on the seashore where the currents of water have gradually concentrated the gold into a richer deposit than the average sand.

2. Deep, or Ancient, Placers. — Placers in some instances have become deep by excessive deposits of material; the rivers that formed them were diverted from their original courses by filling up, by changes in the elevation of the continent, or by lava flows filling the upper portion of their courses. But such placer deposits may have been subsequently cut by the modern rivers, when they will form *bench* or *hill placers*. In some cases the deposits may be

from 400 to 500 feet in thickness, the upper portions being, as a rule, composed of lower grade material than those portions nearer bed rock. In other cases the old river bed may have become covered with lava, to a greater or less depth; subsequent erosion may then have formed new channels in the old deposits. The deep placers, whether covered with lava or not, frequently become cemented into a kind of conglomerate, on account of the presence of oxide of iron, silicious, or calcareous matter, which has been carried into the deposit by percolating waters. The deep or buried placers were first discovered at points where they were intersected by the courses of modern rivers, and were explored by drifts or tunnels.

3. Occurrence of Gold in Placers.—Placer deposits contain metallic gold in fragments ranging from the finest dust to nuggets that weigh several pounds. The gold is sometimes associated with more or less platinum and platinum metals, and occasionally with silver, lead, copper, black magnetic iron sand, tin oxide, and precious stones.

4. Crevices.—The velocity of the current and the amount of material carried by the river will determine whether it will erode its channel or deposit material. If the river water holds a small amount of sediment in suspension, erosion will take place; while if it is overloaded, some of the material will be deposited. Just at the point where a river begins to deposit some of the material it is carrying, a concentrating action takes place, and the materials having the greatest density will separate first. As the bed rock of most rivers is rough and forms a series of natural pockets and riffles, the best deposits of precious metals are usually found in them. All changes in direction or width of a river or the character of the bed rock and river bottom will influence the deposition of gold.

This is illustrated in Fig. 1 (*a*), (*b*), and (*c*). A pocket at one side of the bed of the stream, which would form a receptacle for gold, is shown in (*a*). In (*b*) and (*c*) a series of crevices *A* formed by the upturned edge of the strata are

illustrated; these would probably contain rich deposits. After a current has commenced to deposit material the operation is so rapid that a stratum will be formed that contains but a small amount of gold compared with that concentrated on the bed rock. If this be succeeded by a period of somewhat more rapid current, a richer stratum will be formed with the gravel previously laid down as a

FIG. 1

bed; or if the current slows down a bed of clay may be deposited, which will later form a floor on which a deposit will be placed. The beds on which later deposits occur are called **false bed rocks,** and frequently a mass of gravel will have a series of rich streaks running through it parallel to bed rock. Rivers that change their courses back and forth through broad sandy beds act in the same way, and often form rich pockets in the material of the bed.

5. Pockets.—Even though the **pockets** on the bed rock are usually rich, the holes below waterfalls are sometimes barren and contain only a small amount of gold. This is accounted for by the manner in which the water plunging over the waterfall would wash everything, coarse or fine, heavy or light, out of the hole, and then by some sudden change the pot hole may be washed full of gravel or other debris, without an opportunity for concentration to take place.

6. Distribution of Placer Gold.—Even where gold is fairly uniformly distributed throughout the material of a deposit, it is usually richer at or near bed rock, hence the different layers of gravel, sand, or other material will have a somewhat different average value. Fig. 2 illustrates this by the figures opposite the different strata, which represent the value per cubic yard of the several strata as obtained, it will be assumed, by a careful series of tests. It is not to be supposed that all cases will be exactly like this, for sometimes the richest deposit may occur high up in the formation on a false bed rock, and at times the surface of the deposit may be entirely barren.

FIG. 2

EXAMPLES OF PLACERS

ALMA PLACER

7. As an example of ordinary placer mining, that of the Green Mountain Company at Alma, South Park, Colorado, is taken. In South Park, at an altitude of 10,000 feet above the sea, is an extensive area of placer ground located on the banks of the South Platte River and extending from the base of Mt. Lincoln to Fairplay, a distance of over 20 miles. This area consists of pebble banks, boulders, gravel, and sand, that slope gently toward the mountainside on both sides of the stream for an average width of about half a mile, the surface being covered with grass and a few spare trees.

Portions of this placer have been worked at Alma and at Fairplay, but the banks are far from being exhausted The principal hydraulic workings are at Alma, where the banks

are thickest, owing to the convergence of the tributary streams at that point. A good supply of water can be obtained during the summer months, when the beds are worked continuously night and day. The gravel and drift material was probably first transported by glaciers, but subsequently it has evidently been washed and worked over again by the streams that flowed from the cañons.

Where a section of the bank is exposed, as at Alma, it exhibits a structure, from the grass roots down to bed rock, similar to that shown in Fig. 3, which may be described as follows: At the top there is first a foot or two of black earth in which there is little gold; below this, a foot or two of clay and pebbles; then a few feet of irregularly bedded streaks of sand formed by eddies and currents and likewise comparatively poor in gold. The remainder of the deposit to bed rock is composed of from 30 to 50 feet of rounded pebbles and boulders varying in size, from a fraction of an inch to a yard or more in diameter, the whole being cemented together with clay, and in places with iron oxide, so that it forms a tolerably tough conglomerate that requires the aid of a pick or stream of water having a good head for its disintegration. The banks continue on both sides of the creek for several miles, but are thickest at Alma opposite the outlet of the Buckskin and Mosquito cañons. Here the banks have been excavated for a long distance from the river, presenting a vertical cliff in some places 70 feet high and about a mile in length. This cliff is cut by narrow channels that have been made by streams of water or flume waterfalls. Some of these cuts are narrow gashes that do not penetrate far into the bank, while others lead through narrow ravines into wide open excavations surrounded by

FIG. 3

high banks, and whose center is occupied by piles of large boulders that have been thrown out of the sluices in the course of the work. Winding through this mass of debris may be seen the remains of the abandoned gravel sluices.

8. The operations in this district are carried on by hydraulicking; that is, the gravel is broken down and the gold concentrated by water. The sluices for carrying the excavated material pass to the river bottom through a narrow ravine, as shown in Fig. 4, and the ends of the sluices are divided into short curved tributaries so as to distribute the tailings over the river bottom. At the Alma mine two sluices pass through a ravine that is over 1,000 feet long, the mine being 200 feet wide and about 70 feet deep.

Fig. 5 illustrates the workings at the head of the sluices, where the method of carrying on the work may be seen. Water is allowed to flow over the bank as flume waterfalls, which can be seen in the background. The disintegration of the material left standing between the gullies cut by these waterfalls is accomplished by means of hydraulicking, and all the material is washed into the gravel sluices. The sluices are lined with wooden-block pavement. After the washing has continued for some time, a number of large boulders usually accumulate at the feet of the waterfalls and above the sluices. When this occurs, the flume that supplies

the water is turned off and the hydraulic nozzles turned on to some other portion of the bank. The men then climb into the pathway of the refuse stream and remove the larger boulders, some of which have to be blasted before they can be handled. The small boulders are loaded on to a stone boat, which is hoisted out of the way by means of a large derrick, which is operated by a 10-foot Pelton waterwheel. Large boulders are attached to the derricks by means of chains and hoisted without the use of the stone boat. After the large boulders have been removed, the streams from the nozzles are once more turned on and the gravel and pebbles

FIG. 5

that had been held back by the boulders are washed into the flumes so as to expose the sandstone bed rock. The cleaners then dig up and shovel into the sluice the rotten surface of the sandstone to such a depth as experience has proved that the gold will be found. They examine all cracks in the rock and scrape or brush out any gold that may be there. After this, the bed rock cuts are advanced and the work of the flume waterfalls and nozzles resumed. Men with long-handled shovels assist in ground sluicing by helping along or removing boulders, thus keeping the water in as definite a channel as possible, so that the work may be effective.

9. Preliminary Operations Necessary.—Before this particular enterprise was undertaken, the ground was prospected and the presence of gold in paying quantities assured by sinking test shafts and by panning the gravel from the surface down to bed rock. The question of water supply was considered and a reservoir constructed with an area of 5 acres and an average depth of 10 feet. The dam is made of gravel and brush, and is provided with timber cribs and gates for the waste way. The ditch from the dam to the mine is about 2 miles long and carries 2,000 miners' inches; it is 12 feet wide, 3 feet deep, and has a grade of 10 feet to the mile. At one place a flume 240 feet long, 6 feet wide, and 3 feet deep is carried on trestles. At the end of this main flume, there is another that rests on rock and runs at right angles to it, from which four ditches are carried to the general workings. From a penstock placed at the end of one of these branch ditches, two pipe lines are carried to the nozzles. The pipes are 22 inches in diameter at the penstock, but taper gradually to 10 inches in diameter at the nozzles, the lines being each approximately 500 feet long. The two nozzles, which have a diameter of about 4 inches, use 200 miners' inches each, the remainder of the water being employed for operating the derrick, or as a flume waterfall.

The two sluices are each 3 feet wide by 4 feet high and are paved with wooden-block riffles 8 inches thick, which reduce the depth by that amount. They are laid on bed rock, which, in places, has been cut down to receive them, and at curves has been raised on the outer side. Each sluice is 4,000 feet long, but most of the gold collects in the first 400 feet; their grade varies from 4 inches to 12 feet in 100 feet. The force of the water is so great in the latter case that boulders weighing 100 pounds are frequently carried entirely through them.

10. Cleaning Up.—At this mine about 2 ounces of quicksilver are supposed to be used for each ounce of gold to be recovered. The **clean ups** which occur at regular

intervals, are performed by first taking out the riffles in sections and washing everything clear. The packing of small stones that was between the blocks, is then removed with 12-tined forks and the floor cleaned. The quicksilver is next removed and washed from the black sand, after which it is strained, the amalgam retorted, and the bullion melted down, preparatory to being sent away.

ROSCOE PLACER

11. As an example of a placer that has been exposed by means of a wing dam, the **Roscoe placer** may be described. This is situated in the cañon of Clear Creek, Colorado, one of the steepest and grandest in the front range of the Rocky Mountains. This cañon is cut through granite rock for a distance of some 40 miles to an average depth of 1,000 feet. About 13 miles from its outlet on the plain the creek divides, one branch leading up to the gold-mining region of Central City and the other to the gold-mining region of Idaho Springs. The main creek receives the drainage of two gold-bearing districts. At Central City, in addition to the gold derived from the veins and rocks direct, the creek brings down flour gold and fine amalgam in the tailings from the old stamp mills, which recovered only about 40 per cent. of the gold in the ore and lost a good deal of amalgam. These tailings have been accumulating for 30 years and are an addition to the values that the placer formerly contained. Miners and prospectors in the past obtained gold from the shallow or surface washings, but the deep underlying bed rock was out of their reach.

At one point the cañon is crossed by several hard quartz and feldspar veins, which formed a natural dam, that was broken through by the creek as its cañon was being eroded. But at some comparatively recent time huge masses of rock have fallen into the creek at this point in such a manner as

to practically dam it and form a waterfall about 30 feet high. Above this natural dam there is a large expanse of ground consisting of placer material and old tailings called the *Roscoe Placer*. The fall of water over the natural dam gave a good dumping ground for the tailings, and the flat area above made it convenient to construct a large flume along one bank of the creek, into which the stream was turned by means of a wing dam, thus exposing the bed of the creek with its placer ground and accumulation of tailings.

12. Preliminary Work. — The ground having been prospected by sinking shafts to bed rock and the presence of gold assured, a flume 10 feet wide, 6½ feet deep, and 2,400 feet long was constructed on the south bank of the stream capable of carrying all the water of the creek. Its flooring was 4 inches thick and the sides 2 inches, and its grade on curves 1¼ inches to 16 feet, while in straight portions it was ¾ inch to 16 feet. It followed the course of the stream, the outside edge being elevated on curves to prevent the water from splashing. The water was turned into the flume by means of a wing dam, which was first constructed of sand bags and subsequently strengthened by means of a bank of earth and a substantial wooden framework composed of triangular bents. The whole dam was then backed with stone and gravel, and the down-stream face riprapped with boulders taken from the drift material. By this means the entire creek bed above the natural dam was exposed.

To provide a water supply for the hydraulic nozzles and elevators, a dam was constructed about 3 miles up stream and an intake flume 800 feet long built to a combination penstock and sand box. This penstock, which was 8 feet square and 16 feet deep, was provided with a grating that removed the brush, leaves, etc., and was made large and deep enough to serve as a sand box, the surplus water escaping over a weir and flowing back to the river. Part way down, one side of the penstock was connected to a

48-inch wooden-stave pipe made of pine boards banded with round-steel hoops. After leaving the penstock, the pipe was

Fig. 6

buried for a distance of about 300 feet under a stone embankment and then passed under the railroad track by

means of an arch. The wooden pipe extended to about $\frac{3}{8}$ mile from the mine, diminishing gradually from a diameter of 48 inches at the penstock to 22 inches, where it joined a metal pipe. For the last $\frac{1}{8}$ mile there were two metal pipes, one 16 inches and the other 12 inches in diameter, which ran parallel. One of these pipes supplied the water for the nozzle and the other for the hydraulic elevator. The pressure on the pipes was 87 pounds per square inch at the nozzles, which would throw a column of water 4 inches in diameter for a distance of 165 feet. If the pipes were closed at the end, the pressure would be 189 pounds per square inch.

Fig. 7

In addition to the construction of the flume and wing dam for diverting the stream and the pipe lines for supplying water, it was necessary to construct sluices for washing the gravel, and undercurrents that treated the fine material. The main gravel sluice, which was 208 feet long, 4 feet wide, and 3 feet deep, was paved with square, wooden-block riffles. At its end there was a grizzly that removed all material over $\frac{3}{4}$ inch in diameter and passed it to the waterfall, the fine portion falling into a trough, or undercurrent, which was 12 feet wide, 24 feet long, and

set on a grade of 6 inches in 24 feet. This trough was lined with riffles made by nailing narrow wooden slats across the bottom and securing pieces of flat strap iron to the top, so that the edges of the strap iron projected over the slats on both sides. The material that passed over the first undercurrent was taken up by a second sluice and carried to a second undercurrent, which was 24 feet wide, 45 feet long, and covered with burlap or sacking material, which was occasionally drawn off on to wooden rollers and carried to wooden tanks, where it was carefully washed.

FIG 8

The material caught by this undercurrent consisted largely of flour gold and pyrites that had accumulated from the tailings of the mills up the stream. Before the material passed on to the second undercurrent, it passed over a perforated plate, which removed everything over $\frac{1}{8}$ inch in diameter.

13. Method of Operation.—After the flumes and pipe lines were constructed, a pit was sunk at the upper end of the sluice line with the aid of the nozzles and hydraulic elevator, which was of the Ludlum type. Fig. 6 gives a general view of the placer, showing the railroad track on the north bank, the main flume for deflecting the course of the stream on the south bank, and the pit in which operations were carried on in the foreground. To the left

of the pit can be seen the gravel sluices, and below them the two undercurrents. Fig. 7 is a general view showing the entire valley from the point at which water for the pipe line was taken from the stream down to below the last undercurrent. Fig. 8 shows the pit or excavation at the lower end of the workings. In this illustration it will be noticed that water for some of the work is taken directly from the large flumes that deflect the stream, while the water for operating one of the nozzles and the hydraulic elevators comes from the pipe line. Of the two pipes at the back of the illustration, which ascend from the pit into the gravel sluices, one is a gravel elevator and the other a water elevator. The cleaning up, preparing, retorting the amalgam, and melting the bullion are carried on in the ordinary manner.

APPARATUS FOR PLACER MINING

CRADLES

14. Ancient Methods.—Gold was ordinarily washed in a crude manner in all parts of the world, and all the primitive washing apparatus imitated as nearly as possible the methods Nature employed in the formation of placer deposits; that is, the gold-bearing material was subjected to the action of water, so as to wash away the lighter portion and leave the gold behind. These efforts resulted in a number

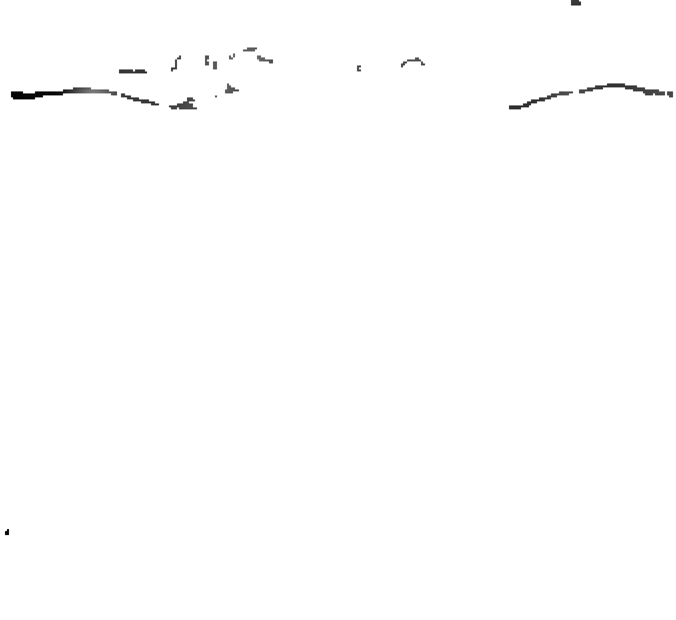


FIG. 9

of washing devices, some of which are used at the present

time and among which may be mentioned the gold pan and batea; the manner in which these are used is shown in Fig. 9.

15. The **cradle** probably originated in Georgia, and was introduced into California during the early days of gold mining; it was used in placer mining before the introduction of sluicing. Cradles are now mainly used in cleaning up placer claims and quartz mills, and for collecting finely divided particles of quicksilver or amalgam.

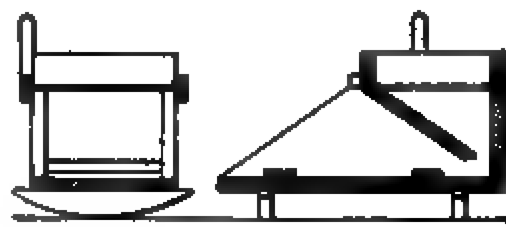


FIG. 10

Fig. 10 gives an end view and a longitudinal section of a cradle. On the upper part there is a removable hopper, 16 inches square inside and about 4 inches deep, with a sheet-iron bottom perforated with $\frac{1}{8}$-inch diameter holes. Beneath the perforated plate, a light frame is placed on an inclination from front to back, on which frame a canvas or carpet apron is stretched. To use the cradle, material is placed into the hopper and water poured on this with a dipper while the cradle is kept rocking. The water washes the fine stuff through the bottom of the hopper, and the gold is either caught on the apron or collects in the bottom of the cradle behind the riffles. The rocks and stones that are too large to be washed through the holes in the plate are picked out from the hopper by hand, washed off, and thrown away. The lighter and worthless material washes over the riffles and discharges at the lower end of the cradle while the gold collects behind the riffles.

FIG. 11

Sometimes the entire bottom of the apparatus under the riffles is covered with carpet. Fig. 11 shows a man operating a cradle.

16. Two-Tray Cradle.—Fig. 12 shows, in section and elevation, a cradle fitted with two trays; it consists of a box, shaped as shown, 40 inches long, 18 inches wide, and 24 inches high. A removable hopper *c* 18 inches by 24 inches by 5 inches, having as a bottom an iron plate perforated with $\frac{3}{8}$-inch holes, is securely fitted on the top. Two handles *d* are attached to the sides of the hopper. Beneath the hopper two removable trays *a* are set at opposite angles. The trays are made of a light frame covered with canvas; sometimes a piece of blanket is used when the gold is very fine, but it is not advisable in other cases, as it is liable to cause trouble by stretching. Two riffles *b* are placed at an angle in the tray to form a pocket. They are usually made of wood but may be formed by sewing a piece of rope across the canvas. As a general rule it is not necessary to put riffles in the bottom of the box. A handle *d* is attached to the side of the box at the point of balance when the cradle is working. If the handle is not at the point of balance the cradle will rock unevenly and will soon break up. Two strong rockers *e* are placed on the bottom of the cradle; from the center of the rocker a bolt projects about 2 inches to fit into an iron plate in the frame on which the rockers work. This keeps the cradle in place.

Fig. 12

The cradle should set at a slight angle so that the water and dirt will run off, the hopper being constructed so that it will be level when the cradle is in place. The whole apparatus should be made strong as the weight and movement of the rocks and dirt produce considerable strain.

17. Washing Trough.—In China, in the Malay Peninsula, and in the Philippine Islands, a kind of shallow trough like a reversed house roof is used for gold washing. This is rocked backwards and forwards in such a manner as to cause the water to flow up one side and then up the other carrying the lighter sand with it. This action gradually works the gold down into the bottom of the trough. The device is easy to handle, but is very slow, it being necessary to repeat the operation several times before fairly clean dust gold can be obtained.

18. The **puddling box** is a wooden box having a series of plugs in the side and may be round, square, or oblong. If round it is usually formed by sawing a barrel in two, though in some cases it may be desirable to use a much larger tub than can be made in this way. The box is filled with water and gold-bearing clay and then stirred, either by means of a rake, or if a round box, by a rotating drag. When the clay has become thoroughly mixed with the water, one of the plugs in the side is removed and the muddy water allowed to run off. After this the plug is replaced, more water is introduced, and the process repeated, with possibly the addition of some more material, until the gravel and black sand carrying gold dust rises to the lowest plug. The contents are then shoveled out and worked, in pans or cradles, by hand. The water from the puddling box is frequently trapped, settled, and used over and over. With round boxes, water can be given a rotating motion that has a tendency to keep the fine material in suspension and thus make a better separation than can be accomplished in the square boxes.

19. The Long Tom.—A rough trough about 12 feet long and from 15 inches to 20 inches wide at the upper end, 30 inches wide at the lower end, and about 8 inches deep is termed a **Long Tom.** It is set on timbers or stones, with an inclination of about 1 inch per foot. The lower end of the box is mitered at an angle of 45° and closed by a sheet-iron plate or riddle perforated with $\frac{1}{8}$-inch holes. Fig. 13 represents the Long Tom *a*, the sluice *d* that feeds it, and the riffle box *b*.

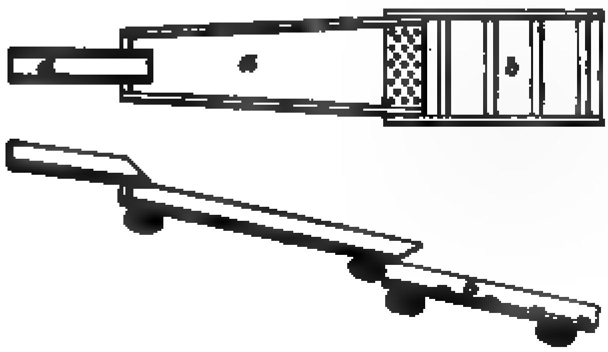


FIG. 13

The material from the sluice flows down over the Long Tom, where it is worked by means of a rake or fork so as to break up the lumps of clay. The finer material passes through the holes on the riddle at the bottom of the Tom,

FIG. 14

while the larger stones collect against it and are removed periodically by means of a shovel or fork. The fine material passes through the riddle into the riffle box, where the heavier

particles, such as gold dust or black sand, collect behind the riffles, while the lighter are washed through. The old-fashioned Long Tom's length was about 14 feet; it was followed by the "Victoria" and the "Jenny Lind," or "Broad Tom." The latter is only from 6 feet to 7 feet in length, 12 inches wide at the upper end and 3 feet at the lower end. From two to four men work at one washer of this class, one man being required to rake and work the material in the portion *a* and discharge the coarse gravel and stones that collect against the riddle, while the others are employed in digging material and shoveling it into the trough *a*. The riffle box *b* is placed at such an inclination that the water passing over it will allow the bottom to remain covered with a thin coating of dirt. Sometimes a little mercury is placed behind the riffles to assist in retaining the gold, and at times the riffle box is supplemented by a series of sluice boxes, which may be provided with blankets in the bottom for catching the very fine gold. Toms are periodically cleaned, the gold and amalgam being collected from the riffles and washed in cradles or pans. Fig. 14 shows three men working at a "Broad Tom."

SLUICING

20. Sluices.—When washing gravel by means of a Tom only comparatively fine sand or gravel goes to the riffle box; on this account quite a good separation can be made with a short series of riffles, but it is usual to add one or two riffle boxes to form a short **sluice**. Owing to the fact that the capacity of the Tom is limited and that the labor involved in washing material and shoveling out the large stones is considerable, this form of apparatus is unsuited for working large quantities of comparatively low-grade material; for which purpose, therefore, sluices were introduced. In the sluice all the material is passed through a series of boxes, and the coarse stones are depended on for grinding and disintegrating the masses of clay, etc.

The term **sluicing** is applied to washing material down any channel, whether it be a box, a ditch dug in the ground on bed rock, or a natural ravine; but when the material is washed over natural surfaces, it is called *ground sluicing* to distinguish it from *box sluicing*. Sluice boxes are made of boards and vary in width from 1 foot to over 5 feet, and in depth from 8 inches to 2 feet, or more; they are usually made in sections from 12 to 14 feet long. For working shallow deposits where the material is shoveled into the sluice, the boxes are ordinarily from 16 to 18 inches wide and from 8 to 12 inches deep; they are frequently constructed slightly narrower at one end than at the other, so that they will telescope one into the other. The advantage of this construction is that the sluice box can be quickly taken up and placed in a different position.

21. Box sluicing is carried on by hydraulicking, manual labor, or mechanical excavators. Fig. 15 shows one method practiced, where the material is shoveled into sluice boxes. The coarse stones are washed in the sluice box and then thrown out, as shown by the pile *b*. In this case only the lower 8 feet of the bank is gold bearing, hence, considerable time and labor must be expended in removing the barren dirt. The pay dirt being quite rich, it pays the miners to follow this method of mining. Larger operators break down the material with streams of water; then scoop up the material in scrapers that receive power from a stationary engine; raise the scraper and contents to a cableway that connects with a tower having sluice boxes in which are riffles for catching the gold.

Fig. 16 shows the method of sluicing just described. The scraper bucket *a*, attached to a wire rope *b*, is raised to the cableway as shown in Fig. 17, and conveyed to the elevated hopper *c*. The contents are dumped into the hopper and washed over screen bars, the fine material passing through into the sluice *d*, while the coarse material passing over the bars into a rock chute *e* accumulates in the pile shown in the illustration. This system is followed with success at

Atlin, British Columbia, and is believed to be original with that district.

22. Grade for Sluices. — Sluices are usually given a uniform grade throughout the whole series; it may vary

Fig. 15

from 8 to 18 inches in 12 feet, depending on the character of the material being washed. It is important in some

cases that the sluice be conveniently near the level of the ground at the point where the pay dirt is introduced, since this has an influence on the grade as well as the pay dirt and the length of the sluice. The steeper the grade the quicker is the dirt washed away by the force of the water, and hence the more material can be washed; tough clay

FIG. 16

does not break up as quickly in a slow current as in a rapid one. In short sluices the grade should be comparatively light, as there is more danger of the fine gold being lost in the short sluices than in the long ones. When working in clay-cemented gravel, the sluices are usually given considerable grade, and numerous steps are also provided to hasten disintegration.

As ordinary pay dirt is generally completely disintegrated in the first 200 feet of a moderately low-grade sluice, the extra length beyond this is useful only for catching the gold. Sometimes, therefore, the grade of the last part of the sluice is reduced. When the grade of the sluice is very low, say 1 foot in 40 or 50, the gold is easily caught, and much of it would rest even upon the smooth floor of the sluice, but additional means, such as riffles, are usually employed.

Fig. 17

Where the grade of a sluice changes, the larger stones are liable to lodge and cause the sluice to overflow, hence particular attention must be paid to this point. For removing stones from the sluice, forks constructed so as to have several prongs are used.

23. Riffles.—If wooden sluice boxes were not lined with false bottoms and sides, the gravel would soon destroy them. The sides are usually lined with planks, but the false bottom is generally in the form of *riffles*, which assist in catching

and holding the gold. **Riffles** are sometimes crevices between the floor lining, at other times they are simply strips of material fastened at right angles to the length of the sluice box, and again they may be formed by leaving spaces between false bottoms, which are constructed with strips of boards placed at right angles above the sluice box floor and to which the false bottom is fastened. The object of riffles is to afford a place where the gold will lodge, and from which it may be recovered. Sometimes the riffles are charged with mercury, which assists in holding the fine gold that might otherwise become dislodged and float away. There is not much danger of losing the coarse gold after it has once been lodged. The specific gravity of mercury is 13.6; ordinary stones will not therefore sink in mercury, but will wash over it; gold has a specific gravity of 19 and platinum 21.5, so that they will sink through the mercury. Compact platinum will not amalgamate with mercury, while gold will, hence it may sometimes be separated from the gold. Silver may be amalgamated by mercury, and fine gold is quickly absorbed, while coarse gold is amalgamated slowly.

24. Bar riffles are composed of bars or slats placed longitudinally in the sluice. They are usually constructed in sections of such a length that it requires two sections to each sluice box; this would make the riffles about 6 feet long. The slats are held apart by small wedges and nailed across the ends to strips. They are held down in the sluice boxes by wedges or by having strips nailed longitudinally above them to the sides of the sluice box. Fig. 18 shows the arrangement of a set of riffle bars. The bars composing the riffles may be made of 2-inch or 4-inch lumber 4 inches or 6 inches wide.

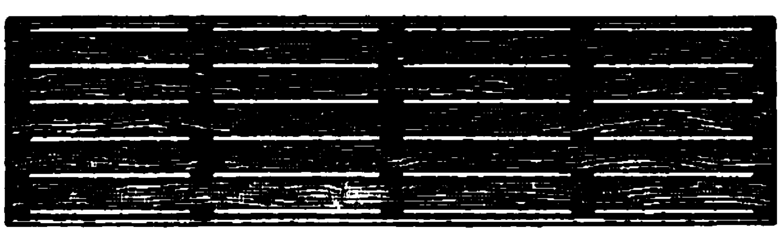


FIG. 18

25. Longitudinal Steel-Shod Riffles.—A form of longitudinal riffle very effective in saving gold is made of

1.5-inch square steel bars about 6 feet long. These are fastened to the edge of planks 1.5 inches thick and 6.5 inches deep, which are set edgewise in the sluice about 1 inch apart, and are held in this position by placing cleats between them at intervals. It is estimated by some that 100 feet of steel riffles are capable of collecting more gold than 1,000 feet of wooden block or rock riffles.

26. Block Riffles.—Where large quantities of pebbles and boulders are passing through the sluice, ordinary longitudinal riffles wear rapidly. To lessen this wear riffles made of wooden blocks sawed across the grain are placed as shown in Fig. 19. These are held in place by having strips of wood nailed to them. By this means narrow

FIG. 19

spaces are left between blocks, which are effective in collecting gold. Sometimes round blocks, Fig. 20, take the place of square blocks, shown in Fig 19. These are held in place by being nailed to transverse strips, as in the case of the square blocks. One series of blocks is started from one side of the sluice and the rest from the opposite side, so that if the sets of blocks do not reach entirely across, the spaces left will not form a channel. The blocks for lining the bottom of

sluices, whether square or round, are usually from 8 to 12 inches high and when worn down to 3 or 4 inches they are discarded.

27. Rock Riffles.— Sometimes sluices are paved with rocks. These riffles may be of squared rocks that have been quarried and dressed approximately like paving stones and then arranged in the sluice in 5-foot or 6-foot sections. Between each transverse row of rocks a piece of timber is securely fastened across the bottom of the sluice, so that in case some of the stones should become loosened, the flow of the material through the sluice cannot rip up the entire series of riffles from one end to the other, and carry the gold caught into the tailings. Another class of rock riffles is formed by simply selecting water-worn boulders or cobble-stones and arranging them as shown in Fig. 19. They are also arranged in sections of about half the length of an ordinary sluice box, with pieces of timber securely fastened between them.

FIG. 20

28. Iron Riffles.—Sometimes the riffles at the head of the sluice are of iron or the entire sluice may be fitted with iron riffles. Fig. 21 illustrates several forms of iron riffles: View (*a*) represents a set of cast-iron riffles that are placed in the sluice so that the material flows in the direction indicated by the arrow. This particular form has been found very effective. In one case 8 feet of such riffles made in sections of about 15 inches each, were placed at the head of a sluice 100 feet long, and caught 98 per cent. of all the gold. These riffles are usually cast in sections 15 or 16 inches long and of the width of the sluice. The succeeding riffles are held in place by plates cast across the end of each section. View (*b*) represents a series of angle-iron

riffles that are secured at the ends by riveting them to a longitudinal strip or by any other simple device, such as placing blocks on the sides of the sluice so as to hold them in place. These riffles have been used in many instances, especially in Australia, New Zealand, and South America. View (*c*) represents a form of cast-iron riffle in which the width of the riffle is equal to the width of the pocket, which

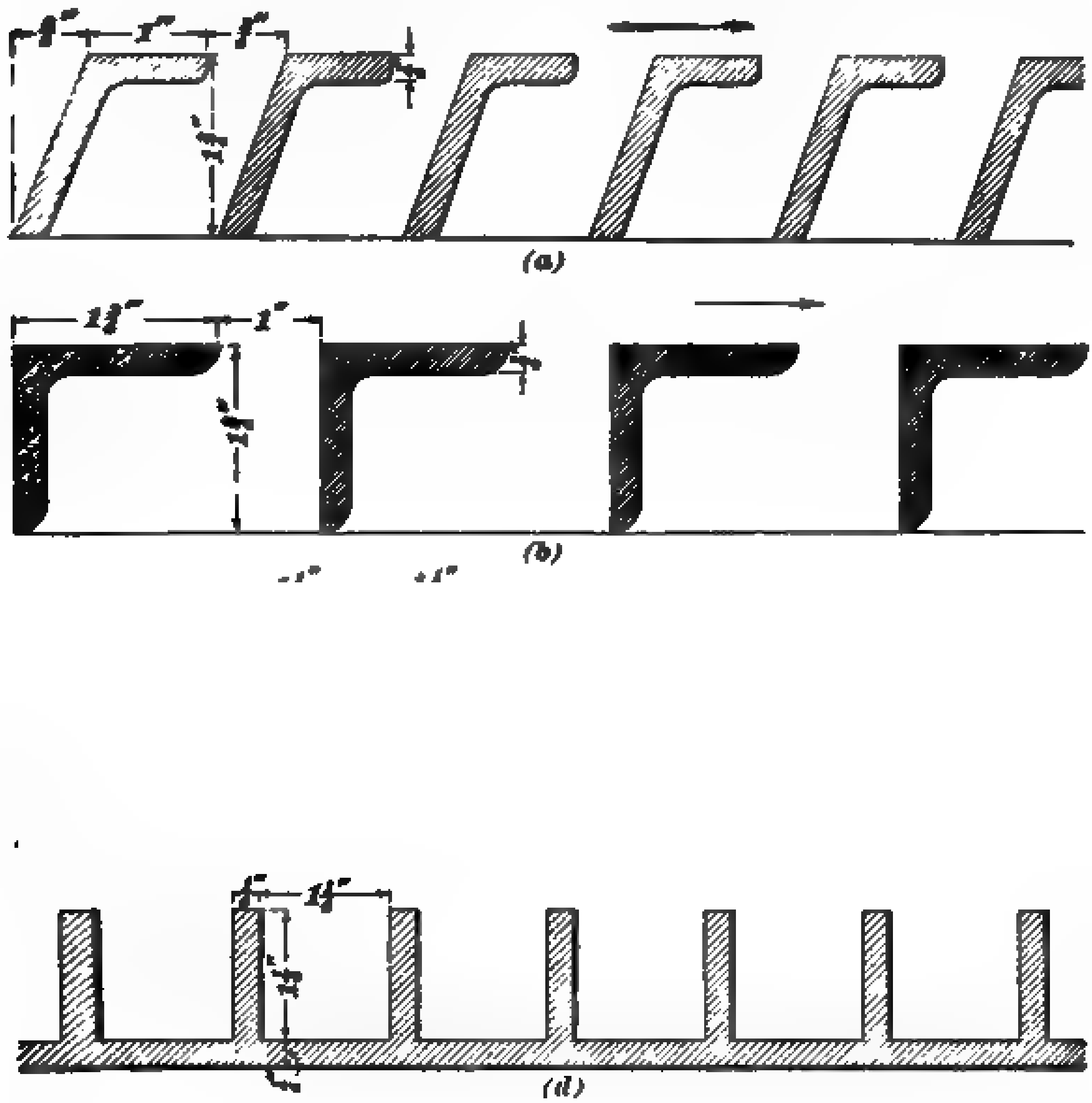


Fig. 21

has vertical sides; the advantage of these riffles is that they are strong and wear well when used in sluices through which large quantities of heavy material pass, but they have not proved to be as close savers as some of the others. View (*d*) shows a riffle whose width is only one-fifth that of its pocket; these riffles have been used in cases where all

the coarse material has been removed from the gravel by means of a trommel, or revolving screen. They have proved to be exceedingly close savers and retain nearly all the gold in the gravel.

29. Rail Riffles.—Where heavy cement gravel is being washed, longitudinal riffles are sometimes made by placing iron rails in the bottom of the sluice. This greatly reduces the wear and tear and forms a good surface for steep portions of sluices where the greatest part of the grinding takes place while the cement gravel is being reduced by means of the boulders in the material.

30. Zigzag Riffles.—Where considerable quantities of fine gold are present in the gravel, **zigzag riffles** are sometimes employed. These extend only partially across the sluice and from alternate sides, so that the current carrying the fine sands is made to pass backwards and forwards from side to side of the sluice. This character of riffles is supposed to assist in amalgamating the fine gold, the amalgam being caught farther down the sluice by means of ordinary riffles.

31. Riffles in General.—In selecting riffles, it will be necessary to consider the character and quantity of the material that is to be passed over them, the amount of water, and the grade of the sluice. Some riffles suitable for dealing with large amounts of coarse material are not as effective as others for the fine material, while some riffles that are especially adapted for fine material are not well suited for coarse material; for instance, the form shown at Fig. 21 (*d*) is intended for fine material, while the forms shown at (*a*), (*b*), and (*c*) are intended for handling coarse or fine material as it may come. Block and stone riffles are about equally efficient with coarse or fine material, but the stone riffles wear longer when large stones are passed through the sluices. Where much of the gold is saved by amalgamation, block riffles made of soft wood seem to be most effective, as they broom up and thus retain the small

particles of amalgam and the gold. Where block riffles are employed, the crevices, or cracks, of the old blocks frequently contain a considerable amount of gold, so that it pays to burn the old blocks and pan the gold out of the ashes. Riffles may be formed in bed-rock sluices by cutting grooves across their floors; these are considered much better than wooden riffles because they wear longer and can, with little trouble, be deepened when they become worn.

32. Undercurrents.—In order to catch the fine gold running through sluices, it is necessary that velocity of current and depth of flow be reduced. To accomplish this, the width of the flume is increased for a distance, while in

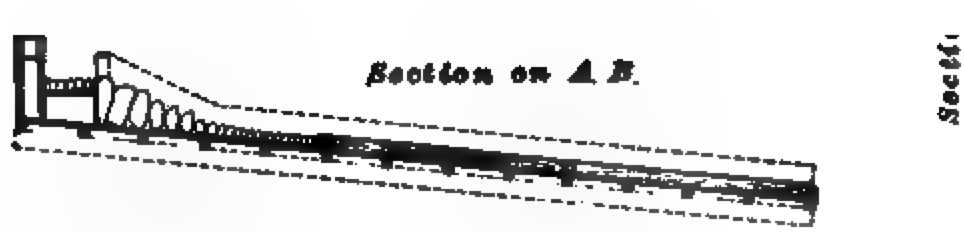


FIG. 22

the bottom of the regular flume is placed a grating, composed of iron bars, set across the flume, from ½ inch to 1 inch apart, as shown in Fig. 22. The fine material drops through between the bars, while the coarse stones and a

portion of the water continue in the main flume. The floor or pavement of the sluice should be at least 1 inch higher than the grating to prevent the latter's clogging. In order to maintain this condition, it is necessary to frequently renew the portion of the pavement immediately before the grating.

In the undercurrent illustrated, after the material passes through the grating at *a*, it flows through a sluice *b* placed at right angles to the regular sluice, and provided with block or cobblestone riffles. This transverse sluice *b* has a slight grade as it recedes from the main sluice, and it also becomes slightly narrower. It is provided with wings or distributing boards (not shown in the drawing) that distribute the flow over the various sections of the undercurrent proper. In the undercurrent shown in the illustration, the two outside sections are paved for a short distance with cobblestones, and for the balance of their length with longitudinal riffles, while the two central sections are paved entirely with cobblestones. Such an undercurrent is intended to handle anything that will pass through the spaces of the grizzly. The grade of the undercurrent may be greater or less than that of the sluice, depending much on the character of the riffles employed.

When there are several undercurrents, the under one may be constructed as shown, or it may have cobblestones, or block or iron riffles throughout, and have a grade somewhat steeper than that of the sluice, the depth of the flow being reduced to 1 or 2 inches. The material that passes over the undercurrent is if possible returned to the main sluice, and frequently the coarse stones that will not go through the grating *a* are passed over a drop in the main sluice, which has a tendency to pulverize any balls of clay or to break up any portions of cement gravel that may have been carried to this point. The next undercurrent would probably be provided with a somewhat flatter grade than the first, and possibly with a different form of riffle. Sometimes the last undercurrent is made much wider than the first, given a very slight grade, sometimes as little as 3 inches

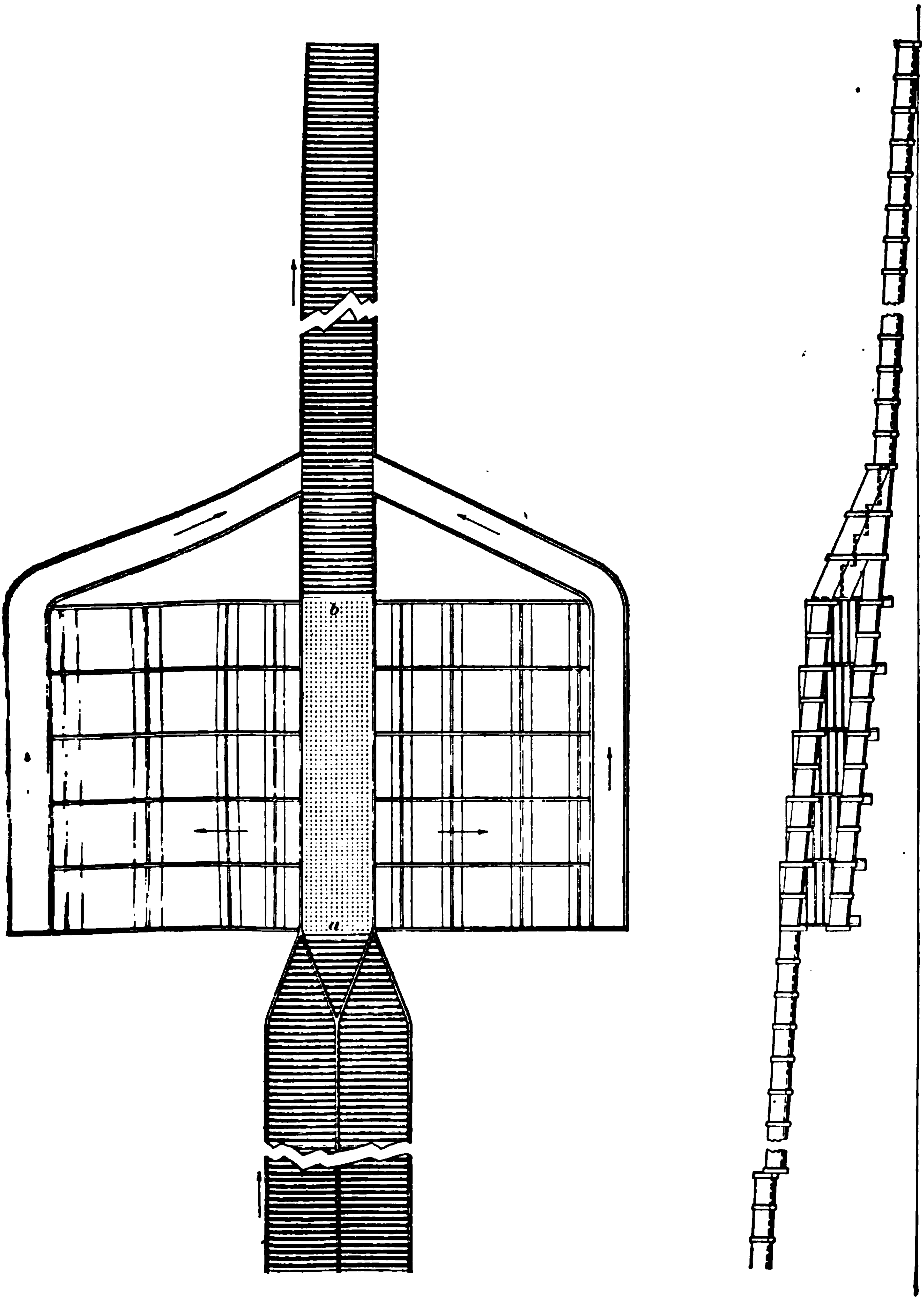


Fig. 23

in 12 feet, and is lined with carpet, blankets, or burlap to collect float gold. The width of the undercurrent is usually from five to ten times that of a sluice, while its length may vary from 20 to 50 feet. It is sometimes possible to arrange an undercurrent at such a place that the coarse stones that pass over the grating are discharged over a precipice, while all the material that passes through is taken to the undercurrent.

33. Gold-Saving Tables.—Undercurrents that are constructed in sections are called **gold-saving tables**; Fig. 23 shows such an arrangement. The sluice above the table is made double, so that either portion of it can be cleaned up separately, or the gates above *a* can be removed and both portions employed at the same time. The main sluice from *a* to *b* has no pavement, but is provided with an iron floor perforated with small holes. The fine material passes through these holes and on to the tables at the right and left, flowing in the direction of the arrows.

These tables may be covered with canvas, carpet, or burlap, or they may be provided with fine riffles; usually they are covered with some fibrous material. After the fine material has passed over the tables, it is taken by the branch sluices to the main sluice lower down. Below the point *b*, the main sluice has a series of steps paved with iron riffles that disintegrate any material that has not been broken above that point. Any section of the tables may be cleaned by turning the material from it on to the adjoining sections by means of deflecting boards under the main sluice.

Gold-saving tables similar to those shown in the illustration are used in South America, Australia, and New Zealand. The riffles are usually made similar to those shown in Fig. 21 (*b*), and the main sluice is lined with cocoa matting before the riffles are put in place; this cocoa matting serves to hold any fine gold that might otherwise escape, and also to prevent currents from flowing under the riffles.

34. Charging the Sluices.—When commencing operations at a placer mine, the sluices are examined to see that they are water-tight. Then water is turned into the pipes and material run through the sluices for a day or more to pack them with sand and gravel. The water is then shut off and a charge of quicksilver put into the upper portion of the sluices. In the case of very long sluices, quicksilver may be put into the first 200 or 300 feet of the boxes and a small quantity distributed through all, except the last 400 feet of the line; in the case of a 6-foot sluice, the first charge may require as much as 3 flasks, or 229.5 pounds. The undercurrents are charged at the same time, and a small amount of quicksilver put into the tailings sluice. Gradually lessening quantities of quicksilver are added daily during the run, the object being to keep the surface of the mercury uncovered and clean on the riffles in the upper portion of the sluice; hence the quantity charged is regulated by the amount exposed to view. In large placer-mining operations two or more tons of quicksilver may be required, a portion being placed in the sluice and the remainder being held in reserve for use as needed. A 24-foot undercurrent may be given from 80 to 90 pounds of quicksilver. In charging the riffles, the quicksilver should not be sprinkled or splashed, as by this action the mercury is reduced to such small particles that they are readily carried away by the swift stream. The surface of the water from mining sluices often yields minute particles of quicksilver, and sometimes float gold.

In the United States the iron flasks in which quicksilver is shipped contain 76½ pounds.

35. Distribution of Gold in Sluices.—In sluicing, the greater part of the gold, usually at least 80 per cent., is caught in the first 200 feet of the sluice. For example, in a claim yielding $63,000 on a 100-days' run, $54,000 was obtained from the first 150 feet and $3,000 from the undercurrents; the second undercurrent, containing 33 per cent. of the gross undercurrent yield, was 78 feet distant and

40 feet below the first. The last undercurrent was 98 feet from the second, with a drop of 50 feet between them; its yield was about $500. The balance of the gold was obtained from the bed rock above the sluices. Another case has been mentioned where 8 feet of iron riffles at the head of a sluice caught 98 per cent. of the gold, the other 2 per cent. being distributed over 92 feet of block riffles.

36. Ground sluicing is much more rapid than manual labor, and has been used in many cases to feed the material into the sluices. The operation consists in permitting a stream of water to flow along the surface of the ground and into the end of the sluice. This stream loosens and carries with it much of the gravel and earth; its action is often assisted by men who stand in the stream or on the bank and loosen the material with pickaxes, bars, or shovels. By this means one man can sometimes run more gravel into the sluice in a day than he would be able to shovel in a month. Ground sluicing naturally partakes of the nature of flume sluicing, and the current tends to sort the material, leaving the boulders, coarse gold, and black sand on the bed rock. The streams of water that flow over the bank and affect the ground sluice are sometimes called *flume waterfalls*.

37. Booming is ground sluicing, on a large scale, by means of an intermittent supply of water. The water is frequently collected behind a dam with an automatic gate, working somewhat like that shown in Fig. 24. When the dam is full of water, the overflow fills the small tank that operates the gate. The weight of the water in the tank opens the gate, so that the entire mass of water

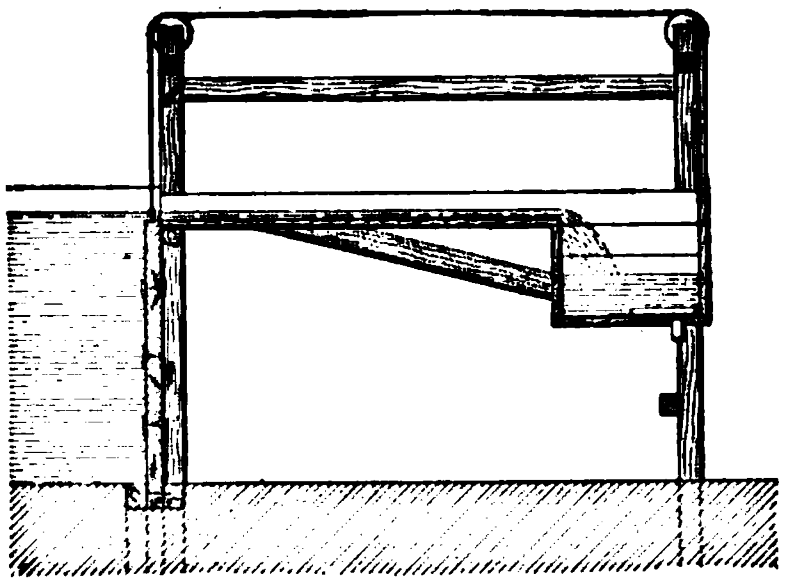


FIG. 24

in the reservoir escapes with a rush and carries the material

before it into the sluices or down through the gulch. The rush of the water lets the heavier particles of gold and the black magnetic iron sand settle on the bed rock or in the sluice; these are subsequently collected and washed in pans or cradles. When the small tank containing the water drops, a trap valve in its bottom is opened, allowing the water in the tank to escape; when this has taken place, the gate closes of its own weight, thus returning the tank to its former position.

38. Wing Dams.—The material below the level of a stream may be worked by means of **wing dams** similar to that shown in Fig. 25. The stream is dammed and the water carried along one side either in a ditch or flume. That portion of the bed separated from the current is pumped comparatively dry by means of hydraulic elevators, power pumps, waterwheels, or Chinese pumps. In Fig. 25, a modified form of Chinese pump is in operation. On the right of the figure will be seen an undershot waterwheel, which operates the pump at the left of the figure. This pump consists of a large wheel around which is passed a chain, which carries the buckets that lift the gravel and water to one of the sluices shown at the left of the figure. The other sluice is the gravel sluice, which is supplied with water from the stream above the dam for washing the gravel taken from the river bed.

FIG. 25

At Oroville, California, head dams and foot dams were constructed 1¼ miles apart, the object being to work the river bed between them for gold. The river, which at this place is from 200 to 300 feet wide, and from 20 to

30 feet deep, had its course changed by means of wing dams; this being accomplished, the water between the dams was pumped out by means of hydraulic elevators. Three such elevators placed 24 feet below the surface, discharged 18,000,000 gallons of water each 24 hours.

HYDRAULICKING

39. The origin of hydraulic mining is usually credited to Edward Mattison, of Connecticut, who was working in the placer mines of California. He conceived the idea of directing a stream of water, under pressure, against the gravel bank from a nozzle, and so doing away with the pick and bar necessary in ground sluicing and flume waterfall work. He conveyed the water through a rawhide hose having a wooden nozzle, and discharged it against the bank in the manner shown in Fig. 26. Others soon took up the method, and the size of the hose and nozzles rapidly increased; but these pipes gave a great deal of trouble by bucking, knocking the men down, and causing considerable damage; stove pipes then replaced the rawhide, and finally the modern piping arrangements with the nozzle known as the giant was perfected.

Fig. 26

40. Water Supply.—The first consideration in hydraulicking is the **water supply**, for on this the mining depends.

The water supply for placer operations may be obtained from running streams, melting snows, and rains. The snow that accumulates on the mountains during the winter and the heavy rains of the spring furnish enormous volumes of water that rush down the gullies and ravines, and, if impounded, can be made to do service in hydraulicking. In case the country is timbered, the streams may furnish a sufficient supply throughout the year; but near the timber line, or in a sparsely timbered district, the water runs off in the spring, leaving the streams dry later in the year. In order to provide a supply of water for hydraulic mining in such a region, it becomes necessary to collect the water in a reservoir, from which it is drawn for use during the dry season.

41. Survey of Water Supply.—To ascertain the quantity of water that will be available in any district for hydraulicking, a survey of the watershed and the measurement of the streams that can furnish a supply are necessary. The next step is to determine a site for a dam, which when constructed will form a reservoir that will contain sufficient water to permit work to be carried on continuously during the summer months. If no such place is available, a series of dams to form storage reservoirs may be constructed, which will answer the same purpose as one large reservoir. The elevation of the proposed reservoir above the mine is next determined, and the least grade that will answer for the flumes ascertained. It is usual to make the pipe line from the flumes to the mine straight, and at the same time obtain all the head of water possible.

MEASURING STREAMS

42. Gauging by V Notch.—A right-angled **V** notch cut from thin sheet iron is sometimes used for gauging small streams of water that will flow through troughs; the notch is fitted in one end of a box, as shown in Fig. 27.

The edge of the plate forming the notch must be sharp, the inside face being at right angles to the surface of the still water, and the bevel, if any, must be away from the water. To prevent surface currents in the box, baffle boards are placed as shown, in the illustration, by dotted lines. These compel the water to flow as shown by the arrows. The distance a of the surface of the water below the top of the notch is taken at least 18 to 24 inches back from the notch, where the water surface is level. This distance subtracted from the total depth H of the notch gives the head h of

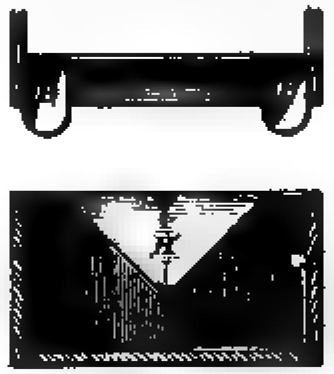


FIG. 27

water passing over the notch. This head may be obtained as follows: A straight edge or level is placed on the weir plate P, so as to extend back over the surface, and the surface of the water is measured. This distance subtracted from H leaves h, which is the depth, or head, of water in the notch. This head may also be obtained by measurements from the bottom of the box, in which case the height of the bottom of the notch above the bottom of the box will be subtracted from the depth of water in the box. This depth may be obtained by measuring with a rule, or scale, which extends to the bottom.

The discharge in cubic feet per second is equal to .0051 times the square root of the fifth power of the head expressed in inches. Table I gives the discharge, in cubic feet per minute, through the right-angled V notch for heads h varying from 1.05 inches up to 12 inches.

TABLE I

DISCHARGE OF WATER THROUGH A RIGHT-ANGLED V NOTCH

h Head Inches	*Q* Quant. Per Min., Cu. Ft.	*h* Head Inches	*Q* Quant. Per Min., Cu. Ft.	*h* Head Inches	*Q* Quant. Per Min., Cu. Ft.	*h* Head Inches	*Q* Quant. Per Min., Cu. Ft.	*h* Head Inches	*Q* Quant. Per Min., Cu. Ft.
1.05	0.3457	3.25	5.827	5.45	21.22	7.65	49.53	9.85	93.18
1.10	0.3884	3.30	6.054	5.50	21.71	7.70	50.34	9.90	94.37
1.15	0.4340	3.35	6.285	5.55	22.20	7.75	51.16	9.95	95.56
1.20	0.4827	3.40	6.523	5.60	22.70	7.80	51.99	10.00	96.77
1.25	0.5345	3.45	6.765	5.65	23.22	7.85	52.83	10.05	97.98
1.30	0.5896	3.50	7.012	5.70	23.74	7.90	53.67	10.10	99.20
1.35	0.6480	3.55	7.266	5.75	24.26	7.95	54.53	10.15	100.43
1.40	0.7096	3.60	7.524	5.80	24.79	8.00	55.39	10.20	101.67
1.45	0.7747	3.65	7.788	5.85	25.33	8.05	56.26	10.25	102.92
1.50	0.8432	3.70	8.058	5.90	25.87	8.10	57.14	10.30	104.18
1.55	0.9153	3.75	8.332	5.95	26.42	8.15	58.03	10.35	105.45
1.60	0.9909	3.80	8.613	6.00	26.98	8.20	58.92	10.40	106.73
1.65	1.0700	3.85	8.899	6.05	27.55	8.25	59.82	10.45	108.02
1.70	1.1530	3.90	9.191	6.10	28.12	8.30	60.73	10.50	109.31
1.75	1.2400	3.95	9.489	6.15	28.70	8.35	61.65	10.55	110.62
1.80	1.3300	4.00	9.792	6.20	29.28	8.40	62.58	10.60	111.94
1.85	1.4240	4.05	10.100	6.25	29.88	8.45	63.51	10.65	113.26
1.90	1.5220	4.10	10.410	6.30	30.48	8.50	64.45	10.70	114.60
1.95	1.6250	4.15	10.730	6.35	31.09	8.55	65.41	10.75	115.94
2.00	1.7310	4.20	11.060	6.40	31.71	8.60	66.37	10.80	117.29
2.05	1.8410	4.25	11.390	6.45	32.33	8.65	67.34	10.85	118.65
2.10	1.9550	4.30	11.730	6.50	32.96	8.70	68.32	10.90	120.02
2.15	2.0740	4.35	12.070	6.55	33.60	8.75	69.30	10.95	121.41
2.20	2.1960	4.40	12.420	6.60	34.24	8.80	70.30	11.00	122.81
2.25	2.3230	4.45	12.780	6.65	34.89	8.85	71.30	11.05	124.21
2.30	2.4550	4.50	13.140	6.70	35.56	8.90	72.31	11.10	125.61
2.35	2.5900	4.55	13.510	6.75	36.23	8.95	73.33	11.15	127.03
2.40	2.7300	4.60	13.890	6.80	36.89	9.00	74.36	11.20	128.45
2.45	2.8750	4.65	14.270	6.85	37.58	9.05	75.40	11.25	129.90
2.50	3.0240	4.70	14.650	6.90	38.27	9.10	76.44	11.30	131.35
2.55	3.1770	4.75	15.040	6.95	38.96	9.15	77.49	11.35	132.81
2.60	3.3350	4.80	15.440	7.00	39.67	9.20	78.55	11.40	134.27
2.65	3.4980	4.85	15.850	7.05	40.38	9.25	79.63	11.45	135.75
2.70	3.6660	4.90	16.260	7.10	41.10	9.30	80.71	11.50	137.23
2.75	3.8380	4.95	16.680	7.15	41.83	9.35	81.80	11.55	138.73
2.80	4.0140	5.00	17.110	7.20	42.56	9.40	82.90	11.60	140.23
2.85	4.1960	5.05	17.540	7.25	43.30	9.45	84.01	11.65	141.75
2.90	4.3820	5.10	17.970	7.30	44.06	9.50	85.12	11.70	143.28
2.95	4.5740	5.15	18.420	7.35	44.82	9.55	86.24	11.75	144.82
3.00	4.7700	5.20	18.870	7.40	45.58	9.60	87.37	11.80	146.36
3.05	4.9710	5.25	19.320	7.45	46.36	9.65	88.52	11.85	147.91
3.10	5.1780	5.30	19.790	7.50	47.14	9.70	89.67	11.90	149.48
3.15	5.3880	5.35	20.260	7.55	47.92	9.75	90.83	11.95	151.05
3.20	5.6050	5.40	20.730	7.60	48.72	9.80	92.00	12.00	152.64

1 cubic foot contains 7.48 U. S. gallons; 1 U. S. gallon weighs 8.34 pounds.

43. **A weir** is an obstruction placed across a stream for the purpose of diverting the water so as to make it flow through the desired channel. This channel may be an opening in the obstruction itself, and it has been found that when properly constructed and carefully managed, such a weir forms one of the most convenient and accurate devices for measuring the discharge of streams. Many careful experiments have been made to determine the quantity of water that will flow over different forms of weirs under varying conditions. As the result of these experiments, two forms have come into general use, and the amount of flow over either can be determined by simple formulas and coefficients that depend on observed conditions.

A *weir with end contractions* is shown in Fig. 28 (*a*). The notch is narrower than the channel through which the water

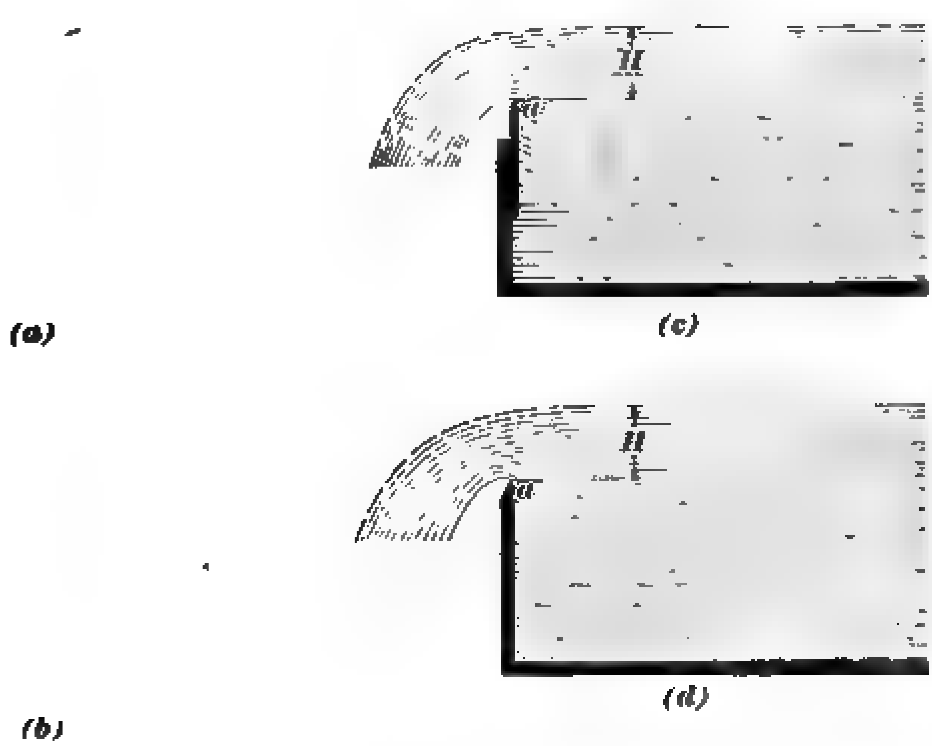


FIG. 28

ordinarily flows, thus causing a contraction at the bottom and two sides of the issuing stream.

A *weir without end contractions*, also called a *weir with end contractions suppressed*, is shown in Fig. 28 (*b*). In this case the notch is the full width of the channel leading to

it, and consequently the stream issuing is contracted at the bottom only.

44. Crest of the Weir.—The edge *a*, Fig. 28 (*c*) and (*d*), is called the **crest of the weir**; it should be beveled so that the water in passing over it touches only a sharp edge. For very accurate work, both side and bottom edges should be made from thin plates of metal having a sharp inner edge, as shown at *a*, Fig. 28 (*c*), but for ordinary work the edges of the board in which the notch is cut may chamfer off, as shown in (*b*). Frequently this edge is not made absolutely sharp, but is left flat for about ⅛ inch, so as to increase the strength of the edge and to decrease the liability of its being damaged. The bottom edge of the notch must be straight and perfectly level; the sides must be at right angles to the bottom. The inside edges of the notch must always be in a plane at right angles to the surface of still water. The head *H* producing the flow is the vertical distance from the crest of the weir to the surface of the water, as shown in Fig. 28 (*c*) and (*d*); this head must be measured at a point sufficiently back from the crest so that the surface of the water is not affected by the curvature of the stream flowing over the weir.

The distance from the crest of the weir to the bed of the stream should be at least three times the head, and with a weir having end contractions, the distances from the vertical edges to the banks of the stream should each be at least three times the head also. The water must approach the weir with little or no velocity; to accomplish which it is sometimes necessary to provide means for reducing the velocity of approach, such as baffle boards similar to those used in connection with the V notch.

45. Plank-Weir Measurements. — Fig. 29 shows a plank weir located in a small stream for the purpose of measuring the flow. The plank dam is put across the stream at a convenient point, care being taken to prevent any leakage around or under the dam. A single board may be used for this purpose, as shown in the figure. A notch

of sufficient depth to pass all the water to be measured, and not more than two-thirds the width of the stream in length is cut in the board and the edges of the notch on the down-stream side beveled within $\frac{1}{8}$ inch, leaving the edge sharp. A stake *b* is driven into the bed of the stream, at a point about 6 feet above the dam until its top is level with the edge *a*, which level may be easily found when the water begins to spill over the top of the board. After the

FIG. 20

water has reached its greatest depth and becomes still at *b*, a careful measurement of its depth over the stake can be made by means of a square, as shown in the figure. In this way a correct measurement may be obtained, whereas if the depth were measured on the notch the curvature of the water would reduce the depth and give an inaccurate result. The latter method, however, is convenient when only an approximate estimate is desired.

In the figure, the dotted line *c* is a level from the bottom of the notch to the top of the stake *b*, while the line *d*

represents the top of the water. The distance between the top of the water and the top of the stake is the true depth, or *spill*, over the weir. The surface of the water below the dam should not be nearer the edge *a* than 10 inches, to insure a free flow over the weir.

46. Method of Using Table II.—When the depth of water over the stake *b* is known, the quantity flowing may be calculated by referring to Table II, which gives the number of cubic feet of water passing per minute over a weir for each inch in breadth, from $\frac{1}{16}$ inch to 25 inches in depth. The figures 1, 2, 3, etc., in the first column are the inches depth of water over the weir, while the first horizontal column represents fractional parts of an inch. The body of the table shows the number of cubic feet and fractional parts of a cubic foot that will pass per minute for a given depth of weir; each result is for 1 inch in width. Thus for any particular number of inches, the result obtained in the table must be multiplied by the number corresponding to the breadth of the weir in inches.

For example, suppose the weir is 30 inches wide, and the water above the stake *b* is $4\frac{1}{2}$ inches deep. The figure 4 is found in the first vertical column, and by following the horizontal line opposite this number until the column headed $\frac{1}{2}$ is reached the number 3.83 is found. This is the number of cubic feet of water that will pass over the weir per minute for 1 inch in width; but since the weir is 30 inches wide, this result must be multiplied by 30, which gives 114.90 cubic feet per minute.

47. Table II may be calculated by the following formula, where

Q = quantity, in cubic feet per minute, for every inch length of weir when contraction of area is reckoned at .62 per cent. of full area, and

H = height of water level over weir in inches, or over the stake *b* in the figure.

$$Q = 4\sqrt{H^3}$$

TABLE II

WEIR TABLE FROM 1-16 INCH TO 25 INCHES DEPTH

Inches		1/16	1/8	3/16	1/4	5/16	3/8	7/16	1/2	9/16	5/8	11/16	3/4	13/16	7/8	15/16
		.006	.01	.03	.05	.07	.09	.11	.14	.17	.20	.23	.26	.30	.33	.36
1	.40	.430	.47	.51	.55	.60	.65	.70	.74	.78	.83	.87	.93	.98	1.03	1.08
2	1.14	1.190	1.24	1.30	1.36	1.41	1.47	1.52	1.59	1.65	1.71	1.77	1.83	1.89	1.96	2.02
3	2.09	2.160	2.23	2.29	2.36	2.43	2.50	2.57	2.63	2.71	2.78	2.85	2.92	2.99	3.07	3.14
4	3.22	3.290	3.37	3.44	3.52	3.60	3.68	3.75	3.83	3.91	3.99	4.07	4.16	4.24	4.32	4.41
5	4.50	4.580	4.67	4.75	4.84	4.92	5.01	5.10	5.18	5.27	5.36	5.45	5.54	5.63	5.72	5.81
6	5.90	6.000	6.09	6.18	6.28	6.37	6.47	6.56	6.65	6.75	6.85	6.95	7.05	7.15	7.25	7.35
7	7.44	7.540	7.64	7.74	7.84	7.94	8.05	8.15	8.25	8.35	8.45	8.55	8.66	8.76	8.86	8.97
8	9.10	9.200	9.31	9.42	9.52	9.63	9.74	9.85	9.96	10.07	10.18	10.29	10.40	10.51	10.62	10.73
9	10.86	10.970	11.08	11.19	11.31	11.42	11.54	11.65	11.77	11.88	12.00	12.12	12.23	12.35	12.47	12.59
10	12.71	12.830	13.95	13.07	13.19	13.31	13.43	13.55	13.67	13.80	13.93	14.04	14.16	14.30	14.42	14.55
11	14.67	14.790	14.92	15.05	15.18	15.30	15.43	15.56	15.67	15.81	15.96	16.08	16.20	16.34	16.46	16.59
12	16.73	16.860	16.99	17.12	17.26	17.39	17.52	17.65	17.78	17.91	18.05	18.18	18.32	18.45	18.58	18.72
13	18.87	19.010	19.14	19.28	19.42	19.55	19.69	19.83	19.97	20.10	20.24	20.38	20.52	20.60	20.80	20.94
14	21.09	21.230	21.37	21.48	21.65	21.79	21.94	22.08	22.22	22.35	22.51	22.65	22.79	22.94	23.08	23.23
15	23.38	23.530	23.67	23.82	23.97	24.11	24.26	24.41	24.56	24.71	24.86	25.01	25.16	25.31	25.46	25.61
16	25.76	25.910	26.06	26.21	26.36	26.51	26.66	26.81	26.97	27.12	27.27	27.43	27.58	27.73	27.89	28.04
17	28.20	28.350	28.51	28.66	28.82	28.98	29.14	29.29	29.45	29.60	29.76	29.92	30.08	30.23	30.39	30.55
18	30.70	30.860	31.02	31.18	31.34	31.50	31.66	31.81	31.98	32.15	32.31	32.47	32.63	32.80	32.96	33.12
19	33.29	33.450	33.61	33.78	33.94	34.11	34.27	34.44	34.60	34.77	34.94	35.10	35.27	35.44	35.60	35.77
20	35.94	36.100	36.27	34.46	36.60	37.87	36.94	37.11	37.28	37.45	37.62	37.79	37.96	38.14	38.31	38.48
21	38.65	38.820	39.00	39.17	39.34	39.52	39.69	39.86	40.04	40.21	40.39	40.56	40.73	40.91	41.09	41.26
22	41.43	41.600	41.78	41.96	42.13	42.31	42.49	42.67	42.84	43.02	43.20	43.38	43.56	43.74	43.92	44.10
23	44.28	44.460	44.64	44.82	45.00	45.18	45.38	45.53	45.71	45.90	46.08	46.26	46.43	46.63	46.81	47.00
24	47.18	47.360	47.55	47.72	47.91	48.09	48.28	48.46	48.65	48.83	49.02	49.20	49.39	49.58	49.76	49.93

If the height H of the water fluctuates during measurement, it will not be correct to take the average of H, but the average of the value $\sqrt{H^3}$.

48. The **miners' inch** originated in California and indicates a method of measurement adopted by the various ditch companies in disposing of water to their customers, and as a unit of water measurement. It is not a definite quantity of water, but varies for the reason that different companies use different heads, and different areas of discharge. The method of measurement formerly employed in Montana was through a vertical rectangle 1 inch high, with a 4-inch head on the center of the orifice. The number of inches was said to be the same as the number of linear inches in the rectangle; thus, under the given head an orifice 1 inch deep and 60 inches long would be 60 miners' inches.

The State Legislature of Montana has now passed a law defining the miners' inch as a certain amount of water flowing per second regardless of the pressure or the size of the opening through which it passes. The statement is as follows: "Where water rights expressed in miners' inches have been granted, 100 miners' inches shall be equivalent to a flow of 2½ cubic feet (18.7 gallons) per second; 200 miners' inches shall be considered equivalent to a flow of 5 cubic feet (37.5 gallons) per second; and this proportion shall be observed in determining the equivalent flow represented by any number of miners' inches." If this amount be reduced to cubic feet per minute, it will be found that a flow of about 1.5 cubic feet per minute is the equivalent of the miners' inch.

49. California Miners' Inch.—In some counties in California what are known as *24-hour*, *12-hour*, *11-hour*, or *10-hour* inches are used to represent the amounts of water that would flow through a given opening in the corresponding lengths of time. A common measurement is through an aperture 2 inches high, and of whatever length is required, in a plank 1¼ inches thick. The lower edge of the

aperture should be 2 inches above the bottom of the measuring box, and the top of the plank 5 inches above the aperture, as shown in Fig. 30, thus making a 6-inch head at the center of the stream. Each square inch of this opening

FIG. 30

represents a miners' inch, which is equal to a flow of 1½ cubic feet per minute. For convenience in converting cubic feet into gallons, it may be stated that a cubic foot of water is equal to 7.48 U. S. gallons.

RESERVOIRS

50. Reservoir Site.—In selecting the **site for a reservoir,** the following points should be observed: (1) A *proper elevation* above the point at which the water is required. (2) The *supply of water* furnished by all creeks and springs, and the catchment area, or area drained into the reservoir, should be carefully determined. In this connection, one must remember that usually all streams and springs are fed by the rainfalls within such area, and hence the total quantity of water can never exceed the rainfalls on the surface,

and owing to evaporation and other losses will always be much less. The *average rainfall and snowfall* should be carefully determined by measurements. The *formation and character of the ground* with reference to the probable amount of absorption and evaporation should be observed.

The elevation of the reservoir depends on the location of the mines and the extent of the country that it is proposed to supply by means of the ditch. The reservoir should be located below the snow line if possible and at the lowest point of the catchment area, or watershed, in order to obtain the maximum supply of water. The average minimum supply of water from all streams should be carefully determined. The rainfall is often greater in the mountain districts than in the lower countries, and is greater on the slopes facing the direction from which the moist winds blow. Snow measurements are taken on the level, and a given amount of snow is reduced to an equivalent amount of water, the total year's fall being calculated as rain.

51. Absorption and Evaporation.—The most desirable ground for a reservoir site is one of compact rock, like granite, gneiss, or slate. Porous rocks, like sandstone and limestone, are not so desirable, since they may absorb water. Steep bare slopes are best, as from them the water will flow directly into the reservoir. The presence of vegetation causes absorption, but at the same time the rainfalls are often greater in regions covered with vegetation, and the streams have a more uniform flow. At the Bowman reservoir in California 75 per cent. of the total rain and snowfall (reduced to rain) of its watershed is said to be collected. A reservoir must be made large enough to hold a supply capable of meeting the maximum demand. The area of the reservoir should be determined, and a table made showing its contents for every foot of depth so that the amount of water available can always be known. Besides the main storage reservoir all hydraulic mines have distributing reservoirs, which receive the water from the main ditch and deliver it to the claims through the pressure pipes. These

auxiliary reservoirs act as safety devices that protect the ditch and flumes from being overflowed, as they are usually capable of holding sufficient water for about a day's run, and if operations should suddenly cease, they are depended upon to take care of the water until the flow in the main ditch can be stopped.

DAMS

52. Introduction.—Dams are constructed for retaining water in reservoirs and for storing debris coming from placer mines in cañons and ravines. The foundation for a dam must be solid to prevent settling, and must be water-tight to prevent leakage under the base, and be arranged in front to prevent wear by water running over the top. Whenever possible, the foundation should be solid rock. Gravel is better than earth, but when gravel is employed, it will be necessary to drive sheet piling under the dam to prevent water from seeping through the formation. Vegetable soil is unreliable, and all porous earths, such as sand and gravel, should be stripped off until hard pan or solid rock formation is reached. In case springs occur in the area covered by the foundation of the dam, it will be necessary to trace them up and confine their flow to the inner or upper side of the dam, so that they will have no tendency to become passageways for water and ultimately wash holes through the foundation and destroy the structure.

53. Wooden dams are constructed of round, hewn, or sawed logs 1 or 2 feet in diameter, laid in a series of cribs 8 or 10 feet square. The logs composing the cribs are pinned together by means of treenails, and the individual cribs are attached by the same means or by bolting. The cribs are usually filled with loose rocks to keep them in place, and in many cases are secured to the bed rock by means of bolts. A layer of plank on the upper face of the dam makes it water-tight.

54. Aprons.—Where water discharges over the top or crest of a dam, it is necessary to provide some surface to receive the impact of the falling water, for otherwise the dam may be undermined and destroyed. If the dam is on firm bed rock, the upper surface can simply be extended slightly and the water allowed to fall on the bed rock, which will not be badly cut; but where there is danger of the foundation being washed away, it will be necessary to provide some form of **apron** or water cushion. An apron may be formed by providing small cribs, which are set on the lower side of the dam and covered with a plank floor, on to which the water falls, and from which it is discharged into the stream below. Sometimes the plank of the floor is made to pitch back toward the dam in such a manner as to form a tank, into which the water falls, the impact of the fall being taken by the water cushion in the tank. A similar result may be accomplished by building a low dam just below the main dam, so as to form a small pond between the two, which acts as a water cushion and protects the bed rock or foundation.

55. Abutments and Discharge Gates.—**Abutments** are the structures at the ends of a dam, and may be constructed from timber, masonry, or dry rockwork. If possible, they should have a curved outline and should be so placed that there is no possibility of the water overflowing them or getting behind them during floods. If the discharge from the dam takes place from the main face, the gates may be arranged in connection with one of the abutments or by means of a culvert through the dam. In either case, there should be some structure above the outlet so as to prevent driftwood, brush, or other material from stopping the discharge gates.

In Fig. 31 the water *a* flows through pipes *b* that are protected by a timber, screen, or strainer *c*. Each pipe has a valve *d* in the culvert *e*, as shown in the illustration, and discharges into the wooden flume *f*, which conveys the water to a flume leading to the mine. When the discharge

gates are placed at one side of the dam, they are usually arranged outside of the regular abutment, between it and

Fig. 31

another special abutment, the discharge being through a series of gates into a flume or ditch.

56. Waste ways are openings provided in dams for discharging surplus water during floods or freshets. In the case of timber dams, they are usually surrounded by solid cribs filled with rocks and have an area of 40 or 50 square feet each. There are two general forms of waste gates. One is a comparatively narrow opening in the dam, extending to a considerable depth, through which water is allowed to discharge during flood time; but when it is desired to stop the flow, planks are placed over the opening in such a manner as to close it. This opening, which is usually not over 3 or 4 feet wide, is provided with guides on the upper face of the dam, between which the planks are slid down, the individual pieces of plank being at least a foot longer than the opening is wide. Another device frequently employed consists in providing a spillway, to one side of the regular spillway, with a crest made of heavy timber 2 or 3 feet lower than the regular crest of the dam. Four or five feet above the crest timbers is placed a parallel timber, the space between which is closed by what are

called *flash boards;* these are made from pieces of 2-inch or 3-inch plank, about 8 or 10 inches wide. The planks are placed against both timbers so as to close the space, and are made sufficiently long to extend from 1 to 2 feet above the upper timber; through the upper end of each plank is bored a hole, through which a piece of rope is passed and a knot tied at the end of the rope, which is secured to staples in the upper timber. When it becomes necessary to open the waste way, men go along with pinch bars and pry up the plank in such a way as to draw the lower end out of contact with the lower timber, when the force of the water will immediately carry it down the stream as far as the rope will allow it to go. After the first plank has been loosened, the succeeding ones can be pulled up with comparative ease, and two men can open a 25-foot or 30-foot section of waste way in a very few minutes. The ropes keep the plank from being lost, and the space can be closed by passing a plank down at one side of the opening, and then moving it sideways in the current. Some skill is required both in opening and closing the waste ways.

57. Dry-Stone Dams.—In regions where cement or lime is expensive and large quantities of suitable rubblestone can be obtained, dams may be constructed without the use of mortar. They are rendered water-tight by a plank facing on the upper side of the dam. Fig. 31 illustrates the main dam for the Bowman reservoir in California; another dam at this reservoir is used as the waste dam. It is supposed that the water will never have to pass over the top of the large dam; but owing to the fact that there are several other reservoirs farther up the stream that might break and flood the reservoir, this dam was constructed so that it would withstand a sudden rush of water over its top. Originally the dam consisted of unhewn cedar and tamarack logs notched and firmly bolted together to form a cribbing that was afterwards solidly filled with loose stones of small size. The original height, 72 feet, was increased to 100 feet by building a dry-stone structure below the main

dam. This is composed mainly of angular stones taken from the mountainside and carefully laid in irregular courses but so that they will break joints. This work was faced with quarried stones on both the upper and lower faces. Part of the lower face was given a batter of 1 inch in every 8 inches of height, and was composed of heavy stones carefully laid and securely bolted together and to the structure behind them. The face of the dam above was also built of quarried stones laid dry. The dam was made water-tight by means of a plank facing on the water side.

If water ever flows over the top of such a dam a great deal of it is liable to pass through the interstices in the slanting stonework, thus subjecting the lower portion of the structure to considerable hydrostatic pressure. To relieve this, the vertical portion of the dam is provided with openings through which this water would find a ready exit.

58. Masonry dams are not much used for placer and hydraulic mining, as the period during which the dams will be required is rarely sufficient to warrant the expense of their construction. Masonry dams are usually lighter than those constructed of dry stone, and their shape is carefully determined according to known laws, on account of the fact that the cement renders stonework one solid mass, while in the case of dry-stone dams, each individual stone in the face of the dam has to resist being washed away by its weight. In dams the stone should not be laid in horizontal courses extending from front to rear.

59. Earth Dams.—Where reservoirs are of moderate depth earth walls may be thrown up to act as dams. Should the material of which an earth dam is constructed not of itself be water-tight, as, for instance, gravel and sand, it is customary to make what is called a **puddle wall.** This consists of a narrow dam made of clay mixed with sand, the two being well tamped or puddled. If the foundation of the dam is gravel or sand, the puddle wall should be carried to bed rock or to an impervious stratum. The puddle wall

should be from 6 to 8 feet wide at the top of the dam, and should be given a slight batter on each side, so that it will be somewhat wider at the base. It is constructed with the dam, and should be protected from direct contact with the water on the upper face by a considerable thickness of earth, to prevent its being eroded before it has had an opportunity to set. The upper face of an earthen dam is usually protected by planks or a stone pavement.

Sometimes earth dams are provided with a core, in place of a puddle wall. This consists of a masonry wall carried to an impervious strata and up through the center of the dam. Such masonry should not be less than 2 or 3 feet thick at the top and should be given a batter of at least 1 foot in every 10 feet high on each side.

DITCHES

60. Introduction.—Thousands of miles of ditches have been excavated in placer-mining districts carrying water from the reservoirs to the mines. On account of the rocky character and uneven surface of the country in such districts, steep grades, high trestles with flumes and wrought-iron or wooden pipe are frequently necessary for conducting the water.

In the construction of ditches, it is sometimes necessary to blast rock, at other times to excavate dirt, in fact all sorts of conditions are met with, some advantageous, others objectionable. After a ditch has been finished and water turned in, it will be found, if the ditch is long, that there will be losses of water from evaporation, filtration, and leakage, and that this loss increases as the length increases.

The loss from evaporation will be greater in summer than in winter, in running than in still water, and in shallow than in deep streams. It is greater in the sun than in the shade, and during winds than in calms, but it does not appear to be greater in hot dry countries than in cooler localities, from the fact probably that in hot countries the air is highly charged with moisture and dews are heavy.

When water soaks into the ground and passes away from a ditch, it may be absorbed by the earth; or if it runs away, form what is known as a leak. The absorption may be considerable in a dry windy region with a hot sun, for earthen banks are then like sponges, which take up water that does not show, since it is evaporated as soon as it reaches the outside of the embankment. Leakage in ditches may be stopped by puddled clay, in a great measure but not entirely. The evaporation, leakage, and absorption on English canals was estimated at 2 inches per day; on the Erie Canal, in New York State, from all causes, 8½ inches per day; on the Pennsylvania canals, at from 4 to 8 inches per day. No definite or average rule can be given for any locality, but when water in a ditch has been running some time and is to be stopped, the losses from various sources can be ascertained by measurements. For example, the full ditch is measured and measurements made every hour until it is dry. These measurements will give the loss per hour.

61. Surveying a Ditch Line.—A preliminary survey of a ditch line can be made by walking and taking elevations with aneroid barometers. By this means the elevations of the termini and intermediate points can be approximately determined. Subsequently surveying parties may be started from these various points for the approximate location of the line. When making the final survey the stations should be properly numbered and staked and pegs driven to grade. Bench marks should be placed at one side of the ditch line so that they will not be disturbed during subsequent work, every ½ or ¼ mile for convenient reference. All details of tunnels, cuts, and depressions requiring fluming or piping should be worked out in full. In this work, the hand level can often be employed with advantage for filling in minor details. Complete notes should be made of the character of the ground along the center line, and also of any possible changes.

62. The **size of the ditch** is regulated by its requirements; its form will be modified, often by circumstances,

according to the judgment of the engineer; half of a regular hexagon is a common form. In a rocky country, narrow and deep ditches with steep grades are adopted in preference to wider ditches with gentler slopes, as they are cheaper to excavate and to keep in repair; hence, ditches with grades of from 16 to 20 feet per mile are quite common in mountainous regions. Before commencing work, the ditch line must be cleared of trees and brush. On a hillside, the line should be graded so that the ditch may have walls of solid, untouched ground, and not mud banks. The banks should be at least three feet wide on top. Where the material excavated is of a comparatively firm nature, the upper side of the ditch in hillside work is frequently given an angle of 60° and the lower bank an angle of 50°; this varies with the nature of the ground. Contracts for digging ditches are either let out per unit of length or at a certain price per cubic yard. The material excavated from the ditch is piled on the lower side and ultimately consolidates into firm ground, thus raising the height of the sides of the ditch and increasing its capacity. If ditches are not so steep as to scour their bottoms, they will ultimately become lined with a layer of fine clay, which closes up the pores and openings in the soil, thus stopping leakage and increasing the carrying capacity of the ditch. It is stated that the annual cost of running and maintaining ditches in California averages $400 per mile. A number of the large California ditches cost nearly half a million dollars each, and some of them are more than 50 miles in length.

Fig. 32 illustrates a cross-section of the North Bloomfield ditch in California. The dimensions of this ditch, as will be seen, were 5 feet in width at the bottom, 8.6 feet at the top, and 3½ feet in depth, with the upper side at an angle of 65° and the lower side at 60°.

FIG. 32

63. Flow of Water in Ditches.—The elements that enter into the flow of water through canals are rubbing surface, area, and grade. The rubbing surface is that section of a canal wet by the water passing through it; hence it is termed the **wet perimeter**; on it depends the quantity of friction or opposition to the flow of water. The wet perimeter that will offer the least opposition to the flow is attained when the width of the bottom is from $1\frac{3}{4}$ to $2\frac{1}{4}$ times the depth of the sides. For example, a ditch having a cross-section whose wet perimeter is 480 square inches, will develop the least amount of friction when the width is to its depth as 30 to 16, or 32 to 15, these dimensions coming within the limits of $1\frac{3}{4}$ and $2\frac{1}{4}$.

The **grade** of a ditch should be uniform to the pressure box, otherwise the current will scour the bottom and sides in one place and deposit the dirt in another place where the grade is not so heavy; the scouring action increases, it is supposed, as the square of the velocity. A current of water moving at the rate of 1 inch per second covers 5 feet per minute, which is equivalent to .057 of a mile, or 300 feet per hour. A current moving 1 foot per second travels 60 feet per minute, that is, .68 of a mile, and in 1 hour will move 3,600 feet. It has been ascertained that when water has a velocity of 1,760 feet, or $\frac{1}{3}$ mile, per hour, it will lift fine sand; and that when moving 2,323 feet, or .44 of a mile, per hour it will lift sand as coarse as linseed.

64. To find approximately what grade must be given a ditch to discharge a given quantity of water in a given time, the area of the ditch and the velocity of the water must be known, then the required fall in inches per mile may be found.

Fall in every foot of length

$$\frac{V^2p + .0001114}{a} + \frac{Vp \times .00002426}{a}$$

V = velocity;

p = wet perimeter;

a = area of waterway in square feet; .0001114 a constant and .00002426 a constant.

EXAMPLE.--What fall per foot must be given a ditch 10 feet wide and having 3.5 feet depth of water to enable it to discharge 102.55 cubic feet per second ?

SOLUTION.— $V = \frac{102.55}{35} = 2.93$ ft. per sec.; $V^2 = 2.93 \times 2.93 = 8.5849$ ft. per sec. $P = 10 + 3.5 + 3.5 = 17$; then

$$\frac{8.5849 \times 17 \times .0001114}{35} + \frac{2.93 \times 17 \times .00002426}{35} = .0005 \text{ ft.}$$

the fall in every foot of length of the canal, or 2.64 ft. per mi.

65. No simple rule can be given for obtaining the area of a ditch that will carry a given quantity of water, and an answer must be sought experimentally. For this purpose assume a convenient section and, the grade being known, calculate its discharge; if this discharge is greater or less than the required discharge, try again with a smaller or larger section until the correct one is found. Another method is to take a ditch having a known area, length, and grade, and by comparison calculate another with the same conditions for data. In actual practice a ditch 5 feet wide at the bottom, 7 feet wide across the surface of the water, discharged 22,000 gallons of water per minute, the water being 3 5 feet deep, with a grade of 8 feet to the mile. The volume of water that will pass through another similarly constructed ditch with the same grade but different area will vary as the square root of the rubbing surface and directly as the area bounded by the wet perimeter.

EXAMPLE.—The area of the above ditch is $\frac{5+7}{2} \times 3.5 = 21$ square feet, what volume of water will flow through a ditch having an area of 15 square feet ?

SOLUTION.—In such cases the perimeter must be assumed for the proposed ditch, for instance $5 + 2.5 + 2.5 = 10$, and for the known ditch it is approximately $5 + 3.5 + 3.5 = 12$. Then

$$\left.\begin{matrix}\sqrt{12} : \sqrt{10} \\ 21 : \phantom{\sqrt{1}}15\end{matrix}\right\} = 22{,}000 : \text{Ans.}; \text{ or } \left.\begin{matrix}3.46 : 3.16 \\ 21 : 15\end{matrix}\right\} = 22{,}000 : \text{Ans., or}$$

$$\frac{3.16 \times \cancelto{5}{15} \times 22{,}000}{3.46 \times \cancelto{7}{21}} = 14{,}350 \text{ gal. per min.}$$

66. Tables of Flow of Water in Open Channels. The following tables are based on the assumption that the waterways are smooth and straight. In them T signifies the top width; B, the bottom width; and D, the depth.

EXAMPLE 1.—The dimensions of a canal are top width 11 feet, bottom width 5 feet, depth 4 feet, and the fall per mile 8 feet. What number of miners' inches will it carry when the proportion of the base to the perpendicular of the side slope is as 3 : 4?

SOLUTION.—In Table III, in column headed Fall Per Mile, find 8 ft., opposite which, in column headed with given specification (11, 5, 4), is found 104.8 cu. ft., the flow per sec. This multiplied by 50, the number of miners' inches equal to 1 cu. ft. flow per sec., gives $104.8 \times 50 = 5{,}240$ as the number of miners' inches required. Ans.

EXAMPLE 2.—Required the number of cubic feet of water that will flow in a canal whose top width is 40 feet, bottom width 20 feet, depth 5 feet, and whose fall is 2 feet per mile, when the proportion of the base to the perpendicular of the side slope is as 2 : 1.

SOLUTION.—In Table III, in column Fall Per Mile, find 2 ft., opposite which, in column headed with the given specifications (40, 20, 5), is found the required flow, viz., 376.1 cu. ft. Ans.

The relative carrying capacity of the trapezoidal form, when the bottom width is to the depth as 5 : 4 and the coefficient of capacity is 1,000, the base : depth of slope = 3 : 4. When the bottom width equals the depth and the coefficient of capacity is .994, the relative carrying capacity of this form is base : depth of slope = 1 : 1. When the sides of a flume are as 2 : 1, the coefficient of capacity is .961; when the channel is semihexagonal in form, it is 1.008; when the channel is square, it is .925; when semicircular, 1.056.

EXAMPLE 3.—The fall being 6 feet per mile, and the sectional area of a square flume 8 square feet, what will be its carrying capacity per second?

SOLUTION.—In Table III, in column of Fall Per Mile, find the given fall 6 ft., opposite which, in column headed Section 8 Sq. Ft., is found 13.65 cu. ft. This multiplied by the coefficient for a square, viz., .925, gives $13.65 \times .925 = 12.63$ cu. ft. Ans.

TABLE III

SHOWING FLOW OF WATER IN OPEN CHANNELS WHEN THE BASE IS TO PERPENDICULAR OF THE SIDE SLOPES AS 8 IS TO 4

Fall Per Mile Feet	Fall Per Rod Inches	T 2.2 Ft. B 1.0 Ft. D .8 Ft. Section 1.28 Sq. Ft. Cu. Ft.	T 3.3 Ft. B 1.5 Ft. D 1.2 Ft. Section 2.88 Sq. Ft. Cu. Ft.	T 4.4 Ft. B 2.0 Ft. D 1.6 Ft. Section 5.12 Sq. Ft Cu. Ft.	T 5.5 Ft. B 2.5 Ft. D 2.0 Ft. Section 8 Sq. Ft. Cu. Ft.	T 6.6 Ft. B 3.0 Ft. D 2.4 Ft. Section 11.52 Sq. Ft. Cu. Ft.	T 7.7 Ft. B 3.5 Ft. D 2.8 Ft. Section 15.68 Sq. Ft. Cu. Ft.	T 8.8 Ft. B 4.0 Ft. D 3.2 Ft Section 20.48 Sq. Ft. Cu. Ft.	T 9.9 Ft B 4.5 Ft. D 3.6 Ft. Section 25.92 Sq. Ft. Cu. Ft.	T 11 Ft. B 5 Ft. D 4 Ft. Section 32 Sq. Ft. Cu. Ft.	T 13.2 Ft. B 6.0 Ft. D 4.8 Ft. Section 46.09 Sq. Ft. Cu. Ft.	T 16.4 Ft. B 7.0 Ft. D 5.6 Ft. Section 62.72 Sq. Ft. Cu. Ft.	T 17.6 Ft. B 8.0 Ft. D 6.4 Ft. Section 81.92 Sq. Ft. Cu. Ft.	T 19.8 Ft B 9.0 Ft. D 7.2 Ft. Section 103.68 Sq. Ft. Cu. Ft.	T 22 Ft. B 10 Ft. D 8 Ft. Section 128 Sq. Ft. Cu. Ft.
1	.0375	.45	1.33	2.67	5.57	9.05	13.46	20.26	28.04	37.1	58.4	96.5	138.3	189.2	261.2
2	.0750	.63	1.88	3.87	7.88	12.80	19.04	28.64	39.67	52.4	82.7	136.4	195.7	267.6	369.4
3	.1125	.77	2.30	4.74	9.65	15.67	23.32	35.08	48.59	64.2	101.4	167.1	239.6	327.7	451.3
4	.1500	.89	2.65	5.47	11.14	18.52	26.93	40.51	56.10	74.1	117.1	192.9	276.7	378.4	522.3
5	.1875	1.00	2.97	6.12	12.46	20.24	30.11	45.30	62.71	82.9	130.9	215.7	309.3	423.1	584.0
6	.2250	1.09	3.25	6.70	13.65	22.17	32.98	49.62	68.70	90.8	143.4	236.3	338.8	463.5	639.8
7	.2625	1.18	3.42	7.24	14.74	23.94	35.63	53.58	74.19	98.1	154.8	255.3	366.0	500.5	691.0
8	.3000	1.26	3.75	7.73	15.75	25.60	38.08	57.28	79.53	104.8	165.5	272.9	391.2	535.1	738.7
9	.3375	1.34	3.98	8.21	16.71	27.15	40.39	60.76	84.14	111.1	175.6	289.4	415.0	567.6	783.5
10	.3750	1.41	4.19	8.65	17.61	28.62	42.57	64.05	88.68	117.1	185.1	305.0	437.4	598.2	825.9
11	.4125	1.48	4.40	9.07	18.47	30.02	44.65	67.18	93.02	122.9	194.1	319.9	458.7	613.2	866.2
12	.4500	1.54	4.60	9.48	19.30	31.35	46.64	70.65	97.15	128.4	202.8	334.2	479.1	655.4	925.6
13	.4875	1.61	4.78	9.86	20.08	32.63	48.54	73.03	101.13	133.6	211.1	347.8	498.7	682.1	941.7
14	.5250	1.67	4.96	10.24	20.84	33.87	50.38	75.79	104.94	138.7	219.0	360.9	517.5	707.8	977.2
15	.5625	1.73	5.14	10.60	21.57	35.05	52.14	78.44	108.63	143.5	226.6	373.6	535.7	732.8	1011.5
16	.6000	1.78	5.31	10.94	22.27	36.20	53.86	81.02	112.18	148.2	234.1	385.9	553.3	756.7	1044.7
17	.6375	1.84	5.47	11.28	22.96	37.31	55.51	89.51	115.64	152.4	241.3	397.8	570.3	780.1	1076.9
18	.6750	1.89	5.63	11.60	23.63	38.39	57.11	85.93	118.99	157.2	248.3	409.3	586.9	802.7	1108.1
19	.7125	1.94	5.78	11.92	24.28	39.44	58.58	88.29	122.26	161.5	255.1	420.5	601.5	824.8	1138.4
20	.7500	1.99	5.93	12.23	24.91	40.47	60.21	90.58	125.43	165.7	261.7	431.4	618.5	846.1	1168.0
21	.7875	2.04	6.08	12.54	25.53	41.47	61.70	92.82	128.53	169.8	268.2	442.0	633.9	867.0	1196.8
22	.8250	2.09	6.22	12.83	26.12	42.45	63.15	95.00	131.55	173.8	274.5	452.5	648.8	887.4	1225.0
23	.8625	2.14	6.36	13.12	26.71	43.40	64.57	97.15	134.51	177.7	280.7	462.9	663.4	907.4	1252.6
24	.9000	2.18	6.50	13.40	27.29	44.34	65.95	99.23	137.40	181.5	286.7	472.6	677.7	926.0	1279.5
25	.9375	2.23	6.63	13.68	27.98	45.24	67.32	101.28	140.24	185.3	292.6	482.3	691.6	946.0	1306.0

TABLE IV

SHOWING FLOW OF WATER IN OPEN CHANNELS WHEN THE BASE IS TO PERPENDICULAR OF THE SIDE SLOPES AS 2 IS TO 1

Fall Per Mile Feet	Fall Per Rod Inches	T 6 Ft. B 2 Ft. D 1 Ft. Section 4 Sq. Ft. Cu. Ft.	T 9.0 Ft. B 3.0 Ft. D 1.5 Ft. Section 9 Sq. Ft. Cu. Ft.	T 12 Ft. B 4 Ft. D 2 Ft. Section 16 Sq. Ft. Cu. Ft.	T 16.0 Ft. B 6.0 Ft. D 2.5 Ft. Section 27.5 Sq. Ft. Cu. Ft.	T 22 Ft. B 10 Ft. D 3 Ft. Section 48 Sq. Ft. Cu. Ft.	T 28 Ft. B 12 Ft. D 4 Ft. Section 30 Sq. Ft. Cu. Ft.	T 40 Ft. B 20 Ft. D 5 Ft. Section 150 Sq. Ft. Cu. Ft.
.5	.018750	1.27	3.85	8.63	18.11	8.79	78.2	188.1
.6667	.025000	1.46	4.44	9 96	20.91	44.79	90.3	217.2
.8333	.031250	1.63	4.96	11.14	23.38	50.08	101.0	242.8
1.0000	.037500	1.79	5.44	12.20	25.61	54.86	110.6	266.0
1.2500	.046875	2.00	6.08	13.64	28.68	61.32	123.7	297.4
1.5000	.056250	2.19	6.67	14.96	21.34	67.26	135.7	326.1
1.7500	.065625	2.37	7.19	16.14	33.88	72.57	146.4	351.8
2.0000	.075000	2.53	7.69	17.26	36.22	77.58	156.5	376.1
2.2500	.084375	2.68	8.16	18.30	38.42	82.29	165.9	399.0
2.5000	.093750	2.83	8.60	19.29	40.50	86.72	174.9	420.6
3.0000	.112500	3.10	9.42	21.14	44.36	95.00	191.6	460.7
3.5000	.131250	3.35	10.17	22.83	47.91	102.60	207.0	497.6
4.0000	.150000	3.58	10.87	24.41	51.22	109.70	221.3	531.9
4.5000	.168750	3.79	11.54	25.88	54.33	116.30	234 7	564.2
5.0000	.187500	4.00	12.16	27.29	57.27	122.70	247.4	594.8
6.0000	.225000	4.38	13.31	29.89	62.74	134.40	271.0	651.5
7.0000	.262500	4.73	14.39	32.29	67.79	145.10	292.7	703.6
8.0000	.300000	5.06	15.38	34.52	72.43	155.20	312.9	752.2
9.0000	.337500	5.37	16.31	36.61	76.83	164.60	331.9	797.9
10.0000	.375000	5.66	17.19	38.59	80.99	173.50	349.9	841.1
11.0000	.412500	5.93	18.03	40.47	84.94	181.90	366.9	882.1
12.0000	.450000	6.20	18.74	42.27	88.72	190.10	383.2	921.5

67. Formulas for Flow of Water Through Open Channels.—The following formulas taken from Trautwine will serve to determine approximately the flow of water through ditches and flumes.

v = mean velocity in feet per second;
a = area of waterway in square feet;
f = fall in feet per foot;
w = wet perimeter in feet.

$$v = \sqrt{\frac{a \times 8 \times 8975}{w}} - .1089 \qquad (1)$$

EXAMPLE 1.—A canal of rectangular cross-section is to be 10 feet wide and to have 3.5 feet depth of water. It is to have a uniform fall, or slope, of .246 foot in a length of 492 feet. What will be the mean velocity of the water, and how many cubic feet of water will it discharge per second?

SOLUTION.—The area of the waterway = $10 \times 3.5 = 35$ sq. ft. The fall $.246 \div 492$ ft. (the distance in which it occurs) = .0005 of a ft. fall for every foot of length. The wet perimeter $= a\,b + b\,c + c\,d = 3.5 + 10 + 3.5 = 17$ ft. Substituting in the formula, the following result is obtained:

$$\sqrt{\frac{35 \times .0005 \times 8975}{17}} - .1089 = 2.9258 \text{ ft. per sec.}$$

or the required velocity. The discharge will be the velocity multiplied by the area of the waterway in square feet, or $2.9311 \times 35 = 102.588$ cu. ft. per sec. Ans.

To find the area, it will be necessary to get the average depth by measuring at different points and multiply the result by the width.

$$f = \frac{v^2 \times w \times .0001114}{a} + \frac{v \times w \times .00002426}{a} \qquad (2)$$

EXAMPLE 2.—Using the same data as in example 1, what fall per foot must be given to a rectangular canal 10 feet wide and having 3.5 feet depth of water, to enable it to discharge 102.55 cubic feet per second?

SOLUTION.—The area = $10 \times 3.5 = 35$ sq. ft., and $\frac{102.55}{35} = 2.93$ ft. per sec. equals the velocity. $2.93^2 = 8.5849$. The wet perimeter

$= ab + bc + cd = 3.5 + 10 + 3.5 = 17$ ft. Substituting in the formula

$$\frac{3.5849 \times 17 \times .0001114}{85} = .000465; \text{ and } \frac{2.93 \times 17 \times .00002426}{85} = .000035$$

$.000465 + .000035 = .0005$ ft., the required fall in every foot of length of the canal, the same as in the preceding example. Ans.

68. The **cost of ditches** depends on the character of the ground to be excavated. If a plow and scraper may be used, the ditching will cost about 20 cents per cubic yard; but if the ground is rocky so as to require pick and shovel, it will probably cost from 25 to 40 cents per cubic yard. If the excavation is rockwork, such as drilling and blasting, it will cost about $2.00 per cubic yard.

For a ditch 3 feet wide at the bottom, 4½ feet wide at the top, and 4 feet deep the figures would be for each 10 feet: Plow and scrapers, $1.11; pick and shovel, from $1.38 to $2.22; rockwork, $11.11. The proportional cost will decrease with the area of the ditch.

FLUMES

69. In general, the use of **flumes** is to be avoided wherever possible, for long experience has demonstrated that they are not economical, being too liable to destruction by fire, wind, snowstorms, and decay; hence they are a source of continual expense. There are, however, instances where the formation of the country requires the use of flumes rather than ditches; for example, in cases where the water must be carried along the face of vertical cliffs. Where a ditch is not as economical as a flume, the ground is composed either of very hard or of porous and broken material; likewise where water is scarce and evaporation and absorption great, flumes are preferable.

Flumes are usually set on a steeper grade than is possible for ditches, the grade frequently being as much as from 25 to 30 feet to the mile. This results in an increase in the velocity of the flow, and hence a proportional decrease in the cross-section of the flume.

70. Construction of Flumes.—Before beginning the construction of a flume the bush for at least 10 feet on each side of the line is burned off. Flumes are usually constructed of seasoned pine plank from 1½ to 2 inches thick, from 12 to 16 feet long. The edge joints are battened on the inside with pine strips from 3 to 4 inches wide and ½ thick. The structure is reinforced every 4 feet by a frame consisting of a sill, a cap, and two posts. A flume 4 feet wide by 3 feet deep requires posts and caps 4 inches by 5 inches, sills 4 inches by 6 inches, and stringers 8 inches by 10 inches.

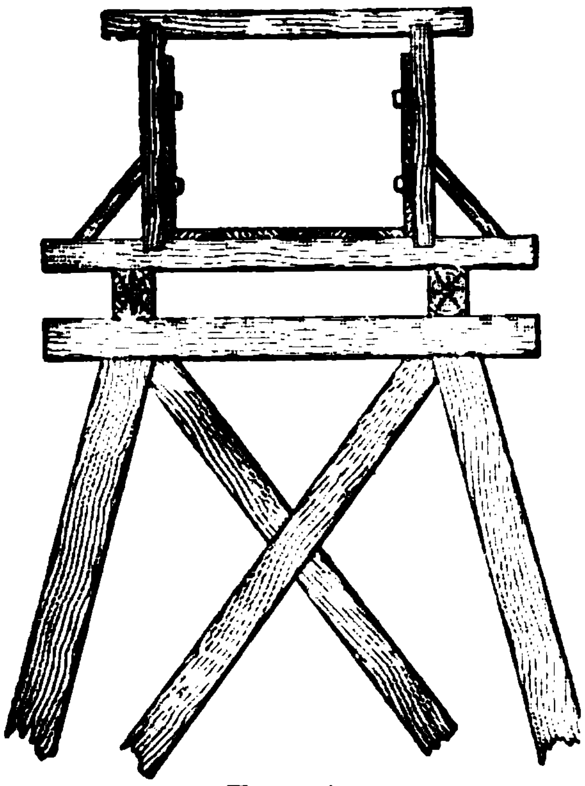


FIG. 33

Posts are set into the sills with the gain 1½ inches deep and are not mortised. The sills are allowed to extend from 12 to 20 inches beyond the posts, and diagonal braces placed as shown in Fig. 33. The posts are usually made sufficiently long to leave a space of 3 or 4 inches between the top of the side planking and the cap. In carrying a flume along a hillside, it should be placed close to the bank, so as to avoid danger from snow slides. The ground for this purpose is first graded and then stringers placed for the flume to rest on. The sills are laid on the stringers to prevent the sills from coming in contact with the earth, and thus protect them from rotting. Another advantage of having a flume close to the bank is that in cold weather the snow usually stops up the space at the side next the bank and prevents the circulation of cold air under it.

Curved flumes should be constructed so as to insure the maximum flow of water and to prevent splashing, as in the latter case excessive freezing is liable to occur in cold weather. The flume is made in sections for curves, thus necessitating more sills, posts, and caps. For good curving, the side planks are sawed partly through in places to make them bend easily. Where the water passes around the

curve, it has a tendency to rise on the outer side of the curve; hence the flume must be blocked up on this side. This is usually accomplished by judging the amount of inclination first and changing it after the water is running by wedging up the flume until all splashing ceases. A fairly good rule for this is to give ¼ inch elevation for each degree of curvature, thus a 12° curve would have 3 inches elevation on its outer side.

In constructing a flume, the sills are placed every 4 feet, the stringers and the bottom planks are nailed to them, the end joints being carefully fitted. The side planks are nailed to the bottom planks and to the posts set in the sills, an occasional cap being placed on the posts to prevent the flume spreading. Sixteenpenny and twentypenny nails are used for fastening the material together. The joints are battened with thin material nailed on with sixpenny nails. Each box when complete is set on grade and wedged into place.

Where a flume connects with a ditch, the posts for a distance of several boxes back are lengthened to permit the introduction of an additional plank on each side. The end boxes of the flume are flared to permit a free entrance and discharge of the water. At the junction of the flume with the ditch, or where a flume passes through the bank of earth, an outer siding may be nailed on the outside of the post to protect the flume. The lumber should be prepared in exact sizes at the mill, so that rapid work can be done in the construction. The lumber is usually delivered at the head of the flume and enough water turned in to float the material down as the work progresses. Where trestles are employed, the bents are usually placed from 8 to 12 feet apart. The life of a flume will usually not exceed 20 years at most, and is generally little more than 10 years.

71. Waste gates should be placed every half mile to empty the flume for repairs or in case of accidents. Waste gates are also useful in running snow out of the flume. In snow belts, flumes are frequently covered with sheds in

exposed places to protect them from snow slides. If anchor ice freezes on the bottom of a flume, the water should be immediately turned out. If snow fills the flume when no water is running through it, it may be gotten rid of by turning on the water and flushing it out before it has time to pack.

72. Bracket Flumes.—When it becomes necessary to carry a flume along the face of a cliff at such an elevation that a trestle is practically out of the question, brackets may be employed. Fig. 34 illustrates a bracket flume employed in Butte County, California. The cliff is a perpendicular wall of basalt, and for a distance of 500 feet the flume is carried on brackets 118 feet above the bed of the ravine, and at one point 232 feet below the top of the cliff. The brackets are

FIG. 34

made of 30-pound **T** rails bent in the shape of an **L**; the longer arm on which the bed of the flume rests is 10 feet long. It is placed horizontally, having the end next the cliff supported in a hole drilled in the rock. The shorter arm stands vertically, and has in its upper end an eye into which is hooked one end of a ¾-inch round-iron rod connecting with a ring bolt soldered into a hole in the cliff above.

The brackets were placed 8 feet apart and were tested to stand a weight of 14½ tons. The flume is 4 feet wide and 3 feet deep with a capacity of 3,000 miners' inches.

73. Cost of Flumes.—The size of the flume, the cost and conveniences for handling material, and the price of labor determine the cost of flumes. It is estimated that with lumber at from $12 to $15 per thousand feet, delivered at the head of the flume so that it may be floated down, the cost for a flume 2.5 feet wide and 2.5 feet high, will be $3.85 per 12 feet; for 6 × 3.5 feet flume, $8.50 per 12 feet.

PLACER MINING

(PART 2)

HYDRAULICKING

PIPES

1. Wooden Pipes.—For moderate pressure due to a head of water, **wooden-stave pipes** are commonly used. While practicable for any desired head, they are economical only to the point where the pressure necessitates such close banding that the cost exceeds that of iron or steel pipe of the same strength. If kept full of water, the stave pipe will last indefinitely, provided the bands are prevented from rusting by a coating of asphaltum or mineral paint. The amount of iron in the bands for each foot of pipe is the same as that required for a foot of sheet-iron pipe of the same diameter calculated, with a considerable margin of safety, to withstand the same head of pressure. Fig. 1 illustrates a wooden-stave pipe in which the bands are composed of round steel rods. One advantage of a wooden-stave pipe is that it can be made to conform to the irregularities of the ground more easily than iron pipe, as will be noticed by the curves in the illustration.

2. Wooden Tunnel Lining.—On some extensive ditch lines it is necessary to carry water through tunnels; owing to the fact that the irregular rock lining of a tunnel interferes considerably with the flow of the water, it has been found best to line the tunnels with lumber. This has been

done by building inside the tunnel the wooden pipe without bands, but backed with cement.

Fig. 1

These tunnels are sometimes driven below the water level of the ditch, so that in case the water should be turned off

in the ditch, the tunnel will always be filled; hence, there will be no tendency for the lining to dry out and crack. Such a lining, always remaining under the water, will last indefinitely.

3. Iron Pipes. — **Wrought-iron** or **steel pipes** are exclusively used for high heads. For low heads, either wood or iron may be employed, the choice between them being a matter of location and cost. Pipes are used to convey water from the pressure box to the nozzles at the mine; also to carry water across depressions. When they are placed in the latter instance so as to follow the natural surface of the ground, they are called *inverted siphons.* The thickness of the metal for pipes is determined by the pressure of the water and the diameter of the pipe. The pipe, when put together, soon becomes water-tight from the foreign matter in the water; this result may be hastened by throwing in a few bags of sawdust or bran. The pipes thus rendered water-tight will remain so when subjected to a pressure as great as 200 pounds per square inch. In the Texas pipe line, Nevada County, California, there is an inverted siphon 17 inches in diameter and 4,438.7 feet long, constructed with riveted sheet iron. Its inlet is 304 feet above the outlet and at the full head will discharge 1,260 miners' inches. The maximum head of this pipe line is 770 feet, which is equivalent to a pressure of 334 pounds per square inch on the pipe at its lowest point.

4. Pipe Joints.—Ordinarily, pipes for hydraulic mining vary from 11 to 40 inches in diameter and are constructed of sheet iron or steel varying in thickness from .12 to .04 inch. The sheets, which are about 30 inches long, are riveted together, and these sections thus formed riveted into lengths of from 20 to 30 feet, or into lengths convenient for transportation.

Sometimes when the pressure is not great they are put together in stovepipe fashion, neither rivets, wire, nor any other contrivance being necessary to hold the joint in place.

Where there is great pressure, iron collars with lead joints are used. Fig. 2 (*a*) shows this latter joint. *f* is a wrought-iron collar about 5 inches in width and $\frac{1}{16}$ inch thicker than the pipe iron; the inside diameter of this collar is $\frac{3}{8}$ inch greater than the outside diameter of the pipe; *l* is a joint composed of lead, which is run in between the collar *f* and the pipe and then calked tight from both sides; *n* is a nipple about 6 inches in length that is riveted in one of the sections by means of $\frac{3}{8}$-inch rivets. Sometimes, owing to the expansion and the contraction of the pipe, the lead in the joint has a tendency to work out; to replace this lead or to force it back into the joint, the clamp shown in Fig. 2 (*b*) has been devised. At *a* is shown the clamp and its method of application for forcing back the lead that has worked out. The clamp is shown in both side view and in cross-section. At the lower part of Fig. 2 (*b*), will be seen another clamp *b*, which is driven over the joint to keep the lead in place after it has been forced in by means of the clamp *a*. Sometimes wrought-iron pipes are provided with hooks riveted near each end to fasten them together by winding wire about the hooks on adjacent pipes, and thus counteract the tendency that pipes have to work apart at the joints, owing to expansion and contraction.

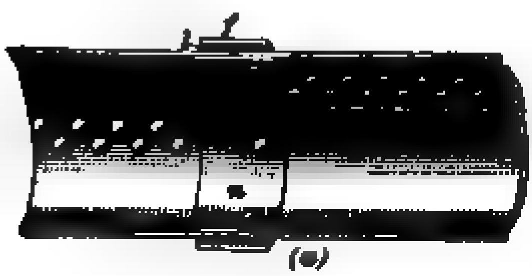

(*a*)

FIG. 2

5. Pipe Elbows. — Sharp bends should always be avoided in pipe lines when possible, and all turns should be made by gradually bending the pipe. When short curves are necessary, elbows similar to that shown in Fig. 3 may be

employed. In this case, *a*, *a* are angle irons riveted to the elbow and connected by straps to similar angle irons riveted

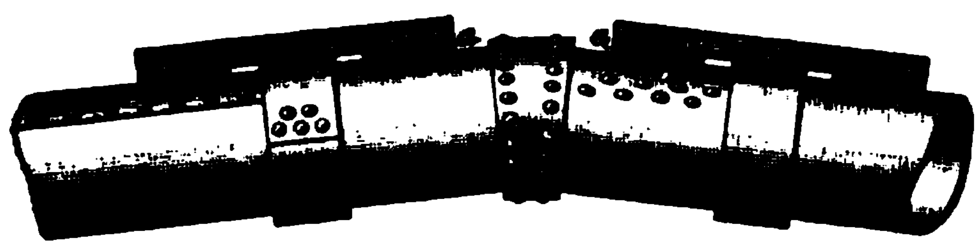


FIG. 3

to the adjacent sections of pipe. These angle irons and straps are necessary to prevent the pipes being pulled apart at this point when contraction and expansion take place.

6. Safety Valves.—Blow-off valves are provided for the escape of air while the pipes are being filled and also to prevent the formation of a vacuum and the consequent collapse of the pipe, which might occur in case of a break. The simplest form is a loaded flap valve of leather on the inside of the pipe, arranged to cover a hole from 1 to 4 inches in diameter.

A very simple automatic valve is shown in Fig. 4, which consists of a small chamber above the pipe, in which hangs an inverted bell, or cylinder *a* closed at the top. When air is escaping, this cylinder will remain in the position shown, owing to its own weight, but as soon as water rises into the chamber, air will be trapped under the bell, causing it to float up and seat against the top of the chamber, thus closing the opening *b* and preventing the escape of the water. Should the flow of water cease, the

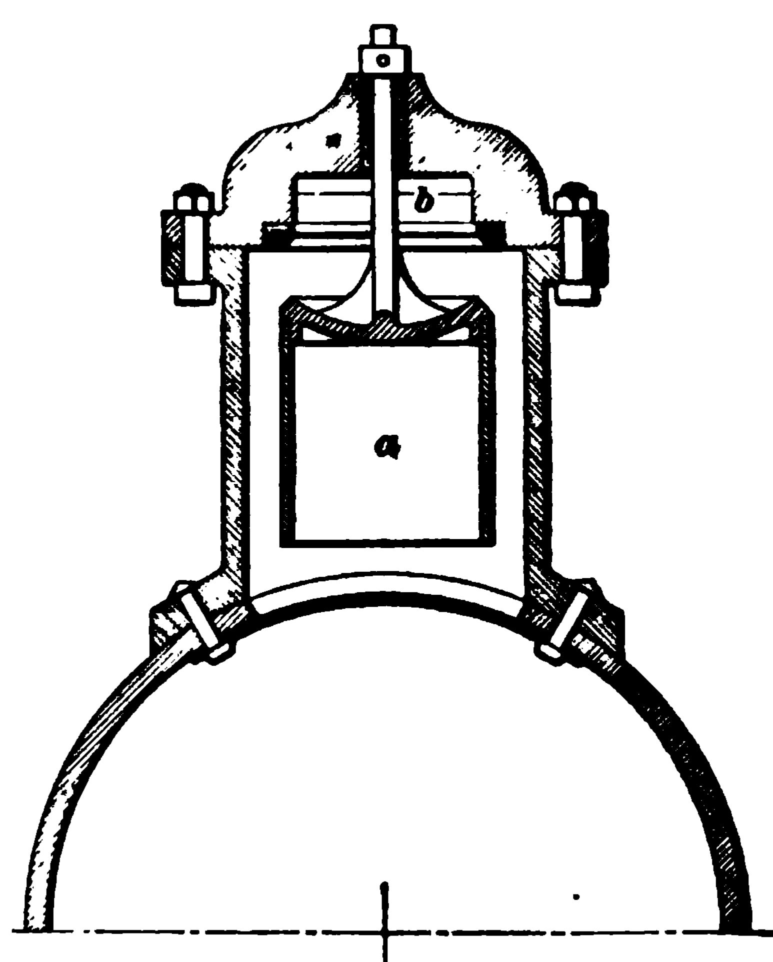


FIG. 4

bell will immediately fall and air will enter through the opening *b*, thus protecting the pipe from collapse.

Fig. 5 shows a form of flow-off, or drain, valve employed at low points for emptying the pipe.

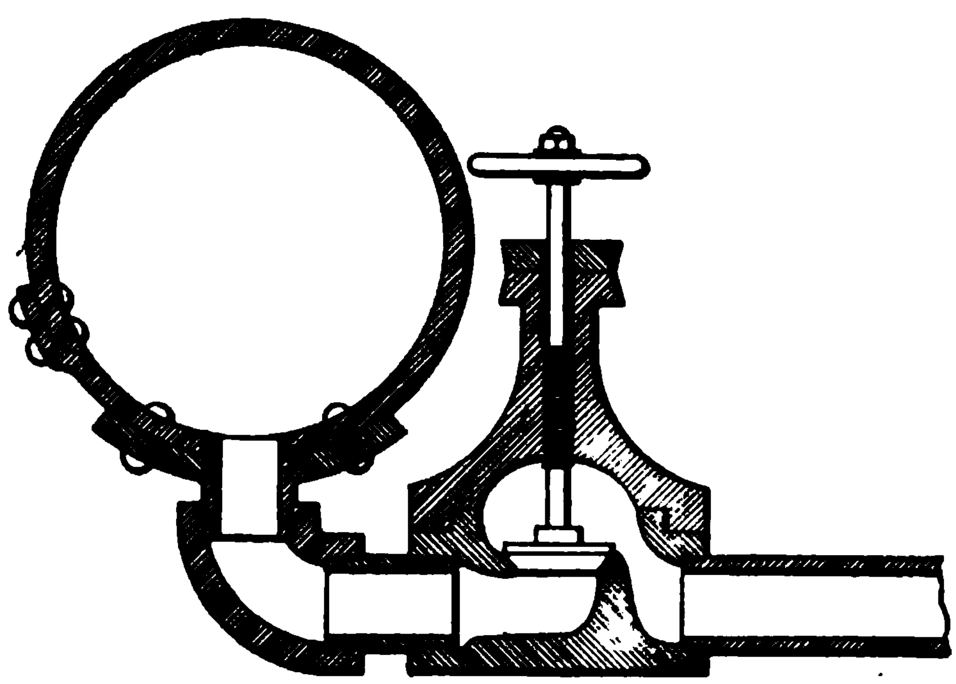


FIG. 5

Fig. 6 shows a combination automatic blow-off and vacuum valve that is employed at high points in the pipe line. The valve on the right is kept closed when the pipe is full and the valve immediately over the pipe open. The pressure in the horizontal tube will keep the central valve closed. In case any small amount of air does collect in the pipe, it can be easily discharged by opening the small valve at the right. If a break should occur anywhere in the pipe line and a vacuum result at the upper point, the central valve would fall of its own weight, thus admitting air and preventing the collapse of the pipe. On refilling the pipe, this valve, being open, allows the air to escape, and when properly constructed will close on being reached by the water. This latter effect is accomplished either by making the lower part of the valve so that it will trap some air and float up; or by shaping the upper disk properly, the escaping water will strike it and lift it high enough so that the current can catch and close it.

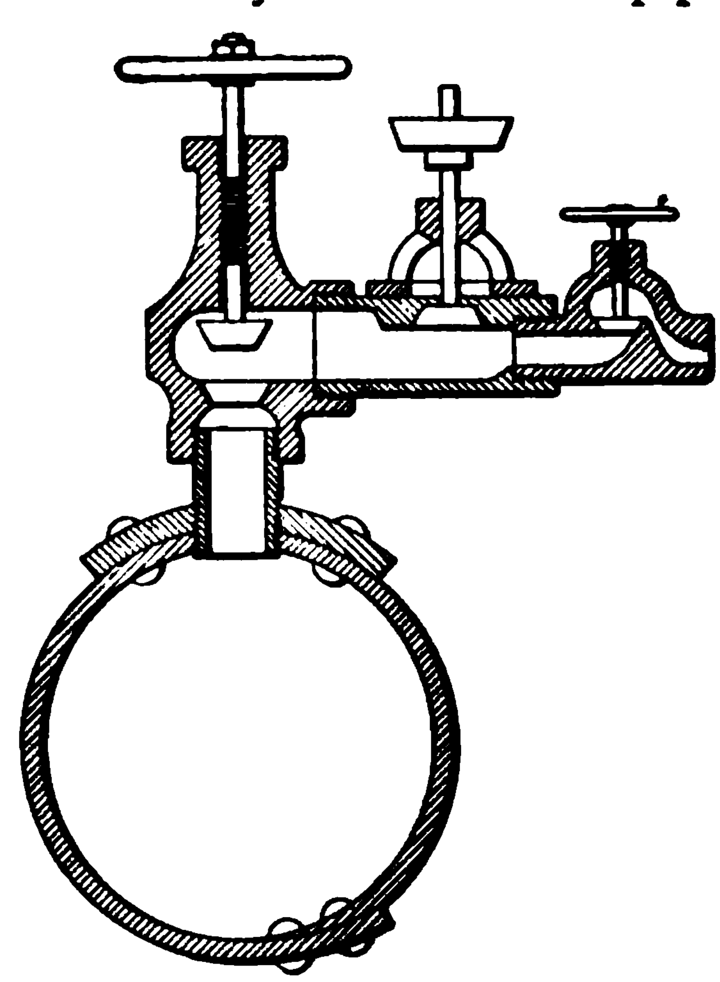


FIG. 6

7. Laying Pipe Lines.—To preserve iron pipe, it should be laid in a trench and covered with earth to a depth of at least 1 foot. Such pipes, well coated with tar or asphalt, have been found in good condition after 15 years of continuous service. The following mixtures have been found to give good results for this purpose: Crude asphalt, 28 per cent.; coal tar (free from oily matter), 72 per cent.; or refined asphalt, 16.5 per cent.; coal tar (free from oily matter), 83.5 per cent.

To prepare either of these, the asphalt is broken into small pieces and heated with the coal tar to a temperature of about 400° F. and well stirred. The pipe to be coated is dried and immersed in this mixture, where it should remain until it acquires the temperature of the bath. When coated, it is removed and placed on trestles to drip and dry in the sun and air. For convenience in immersing, wrought-iron troughs of such a size that they will conveniently contain one section of pipe are provided. Wooden pipe should be painted on the outside with the same mixture that is used for covering the bands and should be covered inside and out with asphalt or coal tar.

8. Filling Pipes. — Pipes should be filled in such a manner as to prevent as far as possible the admission of air, which will be drawn in with the water in surprising quantities unless care is taken. Air escaping from the Fresno pipe nozzle makes a noise that can be heard for a mile or more due to the expansion of air as it leaves the nozzle in bubbles that have been subjected to heavy pressure. The best plan is to put a gate in the pipe below the intake, and thus regulate the flow and maintain a steady pressure. Where the pipes that convey water to the mines are supplied from

FIG. 7

flumes, some kind of box is necessary; this is commonly called a *penstock*, or *pressure box*, and is illustrated in Fig. 7.

A grating of bars should be so placed in the flume as to prevent all dirt or brush from passing into the penstock. The water in the pressure box should be sufficiently deep and quiet to prevent air being carried into the pipe. To accomplish this, the box is frequently made with two compartments, the water flowing from the flume into one and from it to the other through a grating or partition provided with small holes.

As the water coming through ditches almost invariably carries more or less sand, and as this would be liable to cut and scour the inside of the metal pipe, it is quite important that it should be caught in a sand box and not allowed to enter the pipe. The sand box is simply an enlargement in the flume so arranged that the velocity of the current is reduced and the sand allowed to settle from where it is occasionally flushed out by means of a gate near the bottom. Sometimes pressure boxes are made large and provided with a chamber below the intake pipe, it being intended that the sand will accumulate in this chamber and be removed from there periodically.

FIG. 8

9. Supply Pipes. Iron pipes connect the pressure box with the mine and distribute water to the discharge pipes by means of iron gates. The supply pipe is made funnel-shaped to connect with the pressure box, but from there on the main pipe line has a uniform diameter. It should reach the mine by the most direct line possible. In filling the supply pipe, water

should be turned on gradually, for otherwise the moment the pipe becomes filled the sudden check in the flow of the water will result in a violent **water hammer** that may strain the pipe badly, or even burst it. Wherever it is necessary to join the supply pipe and one of the distributing pipes, the present practice is to fork the main pipe by means of a **Y** joint and to provide each branch with a gate valve similar to that shown in Fig. 8.

NOZZLES

EVOLUTION OF THE GIANT

10. Gooseneck.—The original form of nozzle for use on the end of the metal pipe was the **Gooseneck,** shown at

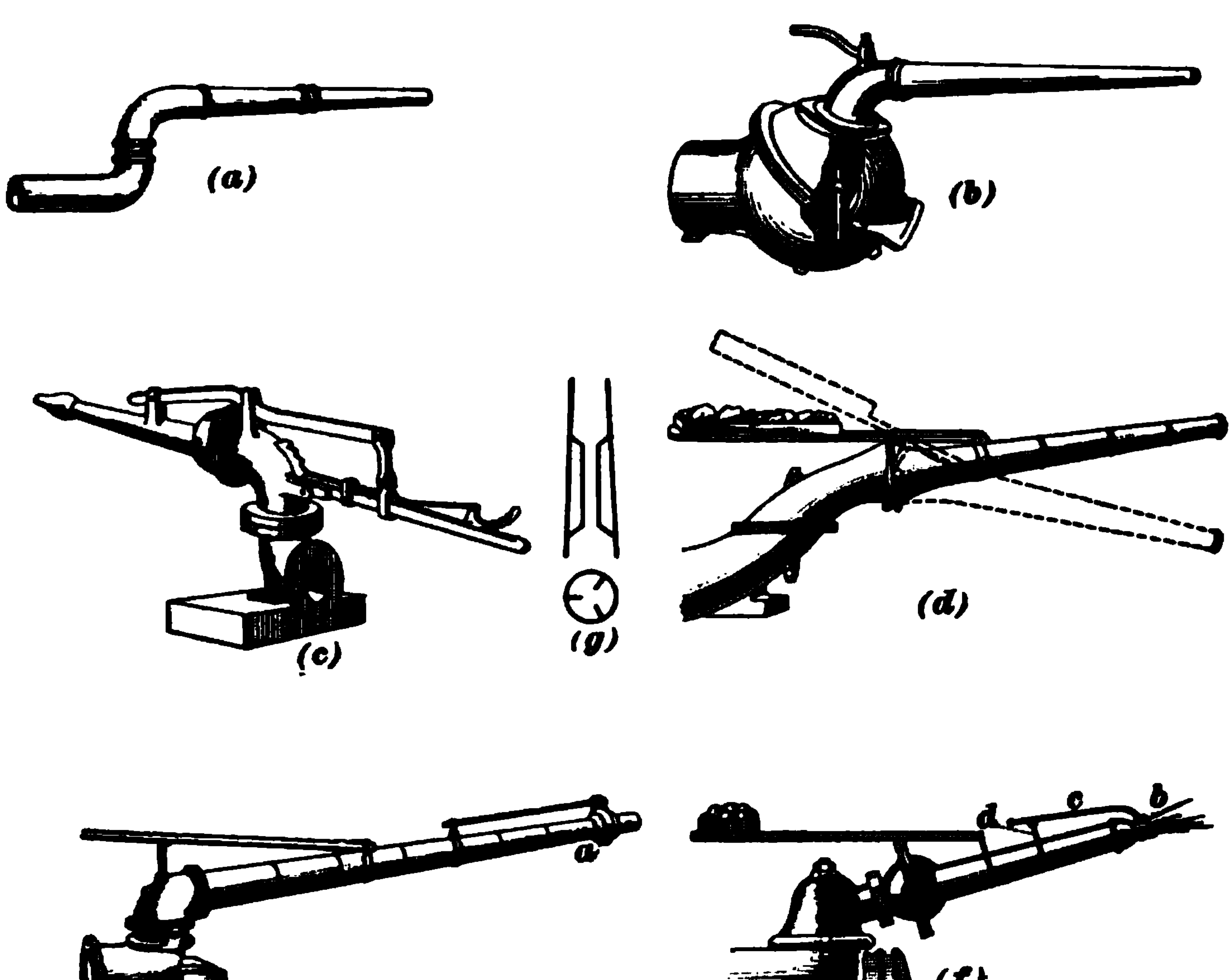


FIG. 9

Fig. 9 (*a*). This consisted of two elbows turned in opposite directions, the upper one being provided with a nozzle. The

pressure of the water caused the joints to leak, and when the nozzle was turned so as to make an angle with the direction of the supply pipe, it would buck, or fly back, thus endangering the lives of the operators.

11. Globe Monitor.—The Gooseneck was succeeded by the Craig Globe Monitor shown at Fig. 9 (*b*). This consisted merely of a ball-and-socket joint, to which the nozzle was attached. The pressure of the water in the joint made it very difficult to operate.

12. Hydraulic Chief.—Mr. F. H. Fisher invented the Hydraulic Chief, which was the next advance. This nozzle is shown at Fig. 9 (*c*). The main improvements consisted in the use of two elbows, placed in reversed positions when in a straight line, connected by a ring in which there were antifriction rollers. The ring was bolted to a flange in the lower elbow but allowed the upper elbow a free horizontal movement, while the vertical motion was obtained by means of a ball-and-socket joint in the outlet of the upper elbow. The interior was unobstructed by bolts or other fastenings, and the pipe man could operate it by means of a lever without danger to himself. Riffles similar to those shown at *g* were inserted in the discharge pipe to prevent the rotary movements of the water caused by the elbows and to force it to issue in a solid stream. These nozzles soon became leaky at the joints, but the riffles have remained as one of the features of the Giants ever since. If the water issues from the nozzle with a rotary motion, it will spray and not form a solid stream.

13. Dictator.—The Hoskins Dictator was the next improvement in nozzles. This was a single-jointed nozzle with elastic packing instead of two metal cases. The joint worked up and down on pivots, and in rotating it small wheels ran around against the flange.

14. Little Giant.—The Little Giant, a subsequent invention of Mr. Hoskins, on account of its simplicity and

durability, superseded all previous nozzles. This is illustrated at Fig. 9 (*d*). It is a two-jointed nozzle, portable and easily handled, having a knuckle joint to give it a vertical range, and a swivel joint in the pipe for the horizontal movement. The nozzle was provided with riffles.

15. Hydraulic Giants.—The Little Giant was succeeded by the Hydraulic Giant, one form of which is shown at Fig. 9 (*e*). In this form, the pivot, or knuckle joint, is placed between the two elbows, the swivel being at the top of the lower elbow. The joint has no bolts passing through it, as was the case in the Little Giant, thus giving a free passage to the water. The pipe is provided with a balance weight similar to that shown in connection with the Little Giant.

16. Monitors.—The Monitor, a nozzle invented by Mr. H. C. Perkins, is shown at Fig. 9 (*f*).

17. Deflecting Nozzle.—In the earlier forms of Giants it was necessary to drag the pipe backwards and forwards, or lift or depress it by manual labor. This was extremely difficult and coupled with more or less danger to the operator. Mr. Hoskins invented the deflecting nozzle, which is a simple device by means of which one man can easily handle the largest Giant. The attachment is shown in connection with the machines illustrated at Fig. 9 (*e*) and (*f*), and may be described as follows: There is a small extension to the regular nozzle, which in some cases is slightly larger, and in others practically of the same diameter as the nozzle. The extension is connected to the main pipe by means of a ball-and-socket joint, as shown at *a*, Fig. 9 (*e*), and at *b*, Fig. 9 (*f*). The deflecting nozzle is operated by means of a handle *c*, which ordinarily rests on a support *d*. When the handle *c* is on the support *d*, the two nozzles are in line. If the deflecting nozzle be moved in any direction by means of the handle *c*, one side of it will be brought into contact with the stream as it issues from the regular nozzle, and

the force of the stream reacting against the deflecting nozzle will force the entire pipe in the direction of the interference. In order to cause the pipe to travel in any given direction, it is only necessary to thrust the handle *c* in that direction. If the pipe man is cool and collected, this deflecting nozzle gives him absolute control over the pipe at all times, and removes much of the danger formerly connected with this part of hydraulic mining.

18. General Remarks on Hydraulic Nozzles.—Monitors or Giants of any description must be securely fastened by being bolted to timbers, the timbers and the first joint of pipe being weighted with rocks; for if the Giant can get into motion and once starts to tremble, it is almost sure to buck, and frequently tears loose from the pipe, so that the Giant and the man at the pipe are both washed away by the flood of water from the supply pipe.

In using the nozzle for hydraulicking considerable skill is required. If the bank contains a stratum of clay, together with a considerable quantity of large boulders, it is possible to cut the clay in such a manner that it will practically lubricate the way for the boulders, and they will slide over the bed rock into the sluices as though the way had been greased. On the other hand, the mixing of the boulders and clay will facilitate the breaking up of clay balls. This is especially true where the sluices are provided with steps. An experienced pipe man, as the man who operates the Giant is called, is much more valuable than several inexperienced men, as he knows just how to take advantage of the different classes of material and to get the most work out of the water.

19. Caving Banks.—In opening up a placer mine, the work is commenced near the sluice. As the bank recedes, bed rock cuts or ground sluices must be advanced to the face to facilitate washing the gravel into the sluice proper. The banks are ordinarily caved by turning two or more streams against any one point in the bank in such a manner

as to undermine it, as illustrated in Fig. 10. When the water is delivered with a force of from 150 to 200 pounds to the square inch, it rapidly undermines the bank, washes away the material, and carries the debris into the sluice. If the bank caves readily, one pipe may be used for cutting, while the stream from the other washes the gravel through the bed rock cuts into the sluices. The face of the bank

FIG. 10

should be kept square, advantage being taken of any corners that may be left, and under no circumstances should a horseshoe shape be formed. When the cut is rapidly pushed ahead and the work not square, the men at the pipes become encircled by the high walls and their lives are in danger. Where the banks exceed 150 feet in height, the deposit is usually worked in two benches. When the men at the pipes see that the bank is about to cave, the water should be

immediately turned away, for if the cave falls on water, a rush of debris is liable to follow that will bury the pipes and force the men to run for their lives. In mines that are only operated by day, caves are usually made just before quitting time at night. Where possible, the washing should be continuous and no water allowed to run to waste; hence it is desirable to have several faces, or openings, so that the stream may be diverted from one to the other while the bed rock cuts, or sluices are being lengthened. Sometimes these cuts are as much as 60 or 70 feet deep. As a precaution against theft, where claims are worked intermittently, the sluices are run full of gravel before closing down.

DISCHARGE FROM NOZZLES

20. Velocity of Efflux.—The vertical height of the water, from the surface in the pressure box to the nozzle, is the *head*, and the velocity with which the water issues from the nozzle follows practically the same laws as those for free falling bodies.

Let

v = velocity of efflux in feet per second;
h = head in feet at the nozzle.

The theoretical velocity of efflux is expressed by the formula

$$v = \sqrt{2gh} = 8.02\sqrt{h}$$

Here $g = 32.16$, that is the velocity of efflux is the same as if the same weight of water had fallen through a height equal to its head. Friction prevents the stream attaining this theoretical velocity.

EXAMPLE 1.—With what velocity will a stream of water issue from a 4-inch nozzle, when the head is 200 feet and the coefficient of velocity is .98 ?

SOLUTION.— $v = 98\sqrt{2gh} = .98\sqrt{2 \times 32.16 \times 200} = 111.15$ ft. per sec.

The quantity of water that will theoretically pass a nozzle will be equal to the cross-sectional area of the nozzle multiplied by the velocity in feet per second.

Let

Q = quantity that passes in 1 second in cubic feet;
A = cross-sectional area of the nozzle in square feet;
v = the mean velocity.

Then

$$Q = A v = .98\, A \sqrt{2 g h} = .98 \times (d^2 \times .7854) \sqrt{2 g h}$$

EXAMPLE 2.—What quantity of water will be discharged from a nozzle 4 inches in diameter, with a head of 200 feet, and a coefficient of velocity of .98?

SOLUTION.— $Q = \frac{12.566}{144} \times 111.15 = 9.7$ cu. ft. per sec. Ans.

The actual discharge is probably not more than 80 per cent. of the theoretical discharge. To reduce this quantity to miners' inches divide the number of cubic feet by 1.5.

21. The Energy of a Jet of Water.—A given weight of water (equal to area of cross-section multiplied by head h) at a velocity v would have an energy expressed by the formula,

$$K = W \frac{v^2}{2g}$$

in which v is the velocity that the water would attain if it fell freely through the height h. The theoretical work that the water can do is equal to its kinetic energy. The quantity of force that a body in motion is capable of exerting when suddenly stopped is called its momentum, but in the case of a stream of water this amount of force is developed at the nozzle only. The force of gravity and the resistance of the air weaken the force of the stream, and at a sufficiently great distance from the nozzle the force of the stream will be entirely dissipated. The above formula will, therefore, only be useful where the momentum can be utilized at the nozzle, as in the case of impulse waterwheels.

EXAMPLE.—What theoretical amount of work, expressed in horsepower per minute, can be obtained from a stream of water issuing

from a 4-inch nozzle, under a head of 200 feet, and with a mean velocity of 111.15 feet per second?

SOLUTION.— W = cu. ft. per sec. multiplied by the weight of 1 cu. ft. of water.

$$9.23 \times 62.5 = 576.9 \text{ lb.};$$
$$v^2 = 111.15 \times 111.15 = 12,354.3;$$

$2g = 64.32$, then

$$K = \frac{576.9 \times 12,354.3}{64.32} = 110,808 \text{ and this divided by 550 ft.-lb. per sec.}$$
$$= 201 \text{ H. P. Ans.}$$

NOTE.—This answer does not agree with the horsepower given in the tables, but is as approximately correct, and on the safe side.

22. Flow of Water Through Nozzles.—The following tables have been computed from a large amount of data obtained by the most careful experiments. They will be found fairly reliable and of much assistance for reference. The velocity in feet for varying heads is given in the second column, Table I, and the cubic feet per minute and horsepower developed under the heading diameter of nozzles. Their use is illustrated in the following examples:

EXAMPLE 1.—The head being 125 feet, (*a*) how many cubic feet per second will a nozzle 4 inches in diameter discharge? (*b*) How many miners' inches?

SOLUTION.—(*a*) In Table I find in the first column the given head 125 ft., opposite which, in column headed 4 in., will be found the required quantity, viz., 7.28 cu. ft. Ans.

(*b*) $7.28 \times 50 = 364$ miners' inches. Ans.

EXAMPLE 2.—Between the inlet and the nozzles of a hydraulic pipe 3 feet in diameter, the distance is 5 miles and the total fall 275 feet. The pipe is to carry 2,000 miners' inches of water, which is to be discharged through two Little Giants, or nozzles equal in size. (*a*) What will be the loss of head by the resistance in the main pipe? (*b*) What will be the size of each nozzle?

SOLUTION.—(*a*) In Table II, find in column headed 36 in. that number which multiplied by 50 will make 2,000, the given number of miners' inches. In this case, 40.86 approximates sufficiently near, opposite which, in the column Fall per Mile, is found 14.78 ft., the loss of head per mile. Multiply this by 5, the length of the pipe, and we have $14.78 \times 5 = 73.9$ ft., the loss of resistance in the pipe 5 mi. long. Subtracting this from the total head, $275 - 73.9 = 201.1$ ft., remaining head. Ans.

TABLE I

QUANTITY AND HORSEPOWER OF FLOW OF WATER THROUGH NOZZLES

Head	Velocity Per Sec.	50 M Inch	100 M Inch	Diameters of Nozzles							
				1 Inch		1.5 Inches		2 Inches		2.5 Inches	
Feet	Feet	1 Cu. Ft. H. P.	2 Cu. Ft. H. P.	Cu. Ft.	H. P.	Cu. Ft.	H. P.	Cu. Ft.	H. P.	Cu. Ft.	H. P.
1.0	8.025	.106	.212	.041	.0046	.093	.010	.164	.018	.255	.029
1.5	9.83	.158	.316	.050	.0085	.111	.019	.200	.034	.312	.053
2.0	11.35	.211	.422	.058	.013	.130	.029	.232	.052	.360	.082
2.5	12.68	.264	.528	.064	.018	.145	.041	.256	.072	.402	.114
3.0	13.90	.317	.634	.061	.024	.159	.054	.284	.096	.440	.150
3.5	15.01	.370	.740	.016	.030	.171	.068	.304	.120	.475	.189
4.0	16.05	.421	.842	.081	.030	.183	.083	.324	.148	.507	.231
4.5	17.02	.474	.948	.086	.044	.194	.099	.344	.176	.540	.275
5.0	17.95	.528	1.06	.091	.051	.205	.113	.364	.204	.56	.315
6.0	19.66	.634	1.27	.100	.068	.224	.153	.400	.272	.622	.425
7.0	21.23	.739	1.48	.108	.086	.242	.193	.432	.344	.672	.535
7.5	21.98	.702	1.58	.111	.095	.250	.214	.444	.380	.697	.595
10.0	25.38	1.06	2.12	.129	.146	.290	.329	.516	.584	.805	.915
12.5	28.37	1.32	2.64	.144	.204	.324	.460	.566	.816	.897	1.28
15.0	31.08	1.59	3.18	.158	.269	.355	.505	.632	1.08	.985	1.68
17.5	33.57	1.85	3.70	.170	.339	.383	.782	.680	1.36	1.06	2.11
20.0	35.89	2.11	4.22	.182	.414	.410	.931	.728	1.66	1.14	2.58
22.5	38.07	2.38	4.76	.193	.494	.435	1.11	.772	1.98	1.21	3.08
25.0	40.13	2.64	5.28	.204	.578	.458	1.30	.816	2.31	1.27	3.61
27.5	42.08	2.90	5.80	.213	.660	.480	1.50	.852	2.60	1.33	4.17
30.0	43.95	3.02	6.04	.228	.760	.513	1.71	.912	3.04	1.42	4.75
32.5	45.75	3.34	6.68	.232	.857	.522	1.93	.928	3.43	1.45	5.35
35.0	47.47	3.69	7.38	.241	.958	.542	2.15	.964	3.83	1.51	5.98
40.0	50.75	4.22	8.44	.257	1.17	.579	2.63	1.03	4.68	1.61	7.31
45.0	53.83	4.75	9.50	.273	1.40	.614	3.14	1.09	5.60	1.71	8.23
50.0	56.75	5.28	10.56	.288	1.64	.648	3.68	1.15	6.56	1.79	10.22
60.0	62.16	6.34	12.68	.385	2.15	.709	4.84	1.26	8.60	1.97	13.43
70.0	67.14	7.39	14.78	.341	2.71	.766	6.10	1.36	10.84	2.13	16.93
80.0	71.78	8.46	16.90	.364	3.31	.819	7.45	1.46	13.24	2.27	20.69
90.0	76.13	9.53	19.06	.386	3.95	.864	8.88	1.54	15.80	2.44	24.68
100.0	80.25	10.56	21.12	.407	4.63	.916	10.41	1.63	18.52	2.54	28.90
125.0	89.72	13.21	26.42	.455	6.47	1.02	14.55	1.82	25.88	2.81	40.40
150.0	98.28	15.85	31.70	.499	8.50	1.12	19.12	2.00	34.00	3.11	53.12
175.0	106.1	18.50	37.00	.539	10.70	1.21	24.07	2.16	42.80	3.36	66.86
200.0	113.5	21.14	42.28	.576	13.1	1.29	29.43	2.30	52.4	3.50	81.75
250.0	127.1	26.62	52.84	.644	18.3	1.45	41.13	2.58	73.2	4.02	114.2
300.0	139.0	31.70	63.40	.705	24.0	1.59	54.07	2.82	96.0	4.40	150.2
350.0	150.1	37.08	74.16	.762	30.3	1.71	68.15	3.05	121.2	4.76	189.3
400.0	160.5	42.27	84.54	.814	37.0	1.83	83.25	3.26	148.0	5.09	231.2
450.0	170.2	47.64	95.28	.864	44.2	1.94	99.34	3.46	176.8	5.40	276.0
500.0	179.4	52.84	105.7	.910	51.7	2.05	116.5	3.64	206.8	5.60	323.2
550.0	188.2	58.22	116.4	.955	59.7	2.10	134.2	3.82	238.8	5.96	372.7
600.0	196.6	63.41	126.8	.999	68.0	2.23	152.9	3.99	272.0	6.23	475.0
700.0	212.3	73.98	148.0	1.06	85.7	2.46	192.8	4.36	342.8	6.79	535.5
800.0	226.9	84.55	169.1	1.15	104.7	2.58	235.5	4.60	418.8	7.19	654.0
900.0	240.7	95.14	190.3	1.22	124.9	2.75	281.0	4.88	499.6	7.63	780.5
1,000.0	253.8	105.6	211.2	1.29	146.2	2.89	329.0	5.16	584.8	8.04	914.0

TABLE I—(*Continued*)

Head Feet	Velocity Per Sec. Feet	150 M Inch 3 Cu. Ft. H. P.	200 M Inch 4 Cu. Ft. H. P.	Diameters of Nozzles 3 Inches		3.5 Inches		4 Inches		4.5 Inches	
				Cu. Ft.	H. P.	Cu. Ft.	H. P.	Cu. Ft.	H. P.	Cu. Ft.	H. P.
1.0	8.025	.308	.424	.372	.040	.50	.056	.656	.072	.81	.09
1.5	9.83	.474	.632	.444	.076	.61	.105	.800	.136	1.00	.17
2.0	11.35	.633	.844	.520	.116	.70	.160	.928	.208	1.17	.26
2.5	12.68	.792	1.06	.580	.164	.79	.224	1.02	.288	1.30	.37
3.0	13.90	.951	1.27	.636	.216	.86	.295	1.14	.384	1.43	.48
3.5	15.01	1.110	1.48	.684	.272	.94	.370	1.22	.480	1.54	.61
4.0	16.05	1.26	1.68	.742	.332	1.02	.452	1.30	.592	1.64	.74
4.5	17.02	1.42	1.90	.776	.396	1.06	.540	1.38	.704	1.74	.81
5.0	17.95	1.58	2.12	.820	.452	1.11	.600	1.46	.816	1.84	1.02
6.0	19.66	1.90	2.54	.896	.612	1.22	.833	1.60	1.09	2.01	1.35
7.0	21.23	2.22	2.96	.968	.772	1.32	1.05	1.73	1.38	2.18	1.74
7.5	21.98	2.38	3.16	1.00	.856	1.36	1.16	1.78	1.52	2.25	1.92
10.0	25.38	3.18	4.24	1.16	1.32	1.57	1.79	2.16	2.34	2.61	2.97
12.5	28.37	3.96	5.28	1.30	1.84	1.76	2.50	2.30	3.46	2.92	4.14
15.0	31.08	4.77	6.36	1.42	2.42	1.93	3.29	2.53	4.32	3.19	5.44
17.5	33.57	5.55	7.40	1.53	3.13	2.08	4.20	2.72	5.44	3.44	7.04
20.0	35.89	6.33	8.44	1.64	3.72	2.23	5.07	2.91	6.64	3.69	8.37
22.5	38.07	7.14	9.52	1.74	4.44	2.36	6.05	3.09	7.92	3.91	9.99
25.0	40.13	7.92	10.56	1.83	5.20	2.54	7.08	3.26	9.24	4.12	11.70
27.5	42.08	8.70	11.60	1.92	6.00	2.61	8.17	3.41	10.68	4.32	13.50
30.0	43.95	9.06	12.08	2.05	6.84	2.79	9.31	3.65	12.16	4.61	15.39
32.5	45.75	10.02	13.36	2.09	7.72	2.84	10.50	3.71	13.72	4.70	17.37
35.0	47.47	11.07	14.76	2.17	8.60	2.95	11.71	3.86	15.32	4.88	19.35
40.0	50.75	12.66	16.88	2.32	10.52	3.15	14.33	4.12	18.72	5.22	23.67
45.0	53.83	14.25	19.00	2.46	12.56	3.34	17.10	4.36	22.40	5.54	28.25
50.0	56.75	15.84	21.12	2.59	14.72	3.52	20.03	4.60	26.24	5.83	32.12
60.0	62.10	19.02	25.36	2.84	19.36	3.86	26.32	5.04	34.40	6.39	43.55
70.0	67.14	22.17	29.56	3.06	24.40	4.17	33.17	5.42	43.36	6.84	54.00
80.0	71.78	25.36	33.86	3.28	20.80	4.40	40.55	5.81	52.96	7.38	67.05
90.0	76.13	28.59	38.12	3.46	35.52	4.73	48.37	6.16	68.20	7.78	79.92
100.0	80.25	31.68	42.24	3.66	41.64	4.98	56.67	6.52	74.08	8.23	93.70
125.0	89.72	39.63	52.84	4.08	58.20	5.57	79.20	7.28	103.5	9.18	130.0
150.0	98.28	47.55	63.40	4.48	76.48	6.10	104.10	8.00	136.0	10.08	172.1
175.0	106.1	55.50	74.00	4.84	96.28	6.60	131.07	8.04	171.2	10.89	216.6
200.0	113.5	63.42	84.56	5.10	117.7	7.05	160.22	9.23	201.3	11.61	261.7
250.0	127.1	79.26	105.7	5.80	164.5	7.88	223.92	10.32	292.8	13.05	370.2
300.0	139.0	95.10	126.8	6.36	216.3	8.63	294.3	11.28	384.0	14.31	486.0
350.0	150.1	111.2	148.3	6.84	272.6	9.33	371.2	12.20	484.8	15.39	613.2
400.0	160.5	126.8	169.1	7.32	323.0	9.97	453.2	13.04	592.0	16.47	749.2
450.0	170.2	142.9	190.6	7.76	397.4	10.58	541.0	13.84	707.2	17.46	894.2
500.0	179.4	158.5	211.4	8.20	406.0	11.15	627.0	14.56	827.2	18.45	1,048.0
550.0	188.2	174.7	232.8	8.40	536.8	11.69	731.0	15.28	955.2	18.90	1,208.0
600.0	196.6	190.2	253.6	8.92	611.0	12.21	832.7	15.96	1,080.0	20.07	1,376.0
700.0	212.3	221.9	296.0	9.84	771.2	13.31	1,051.0	17.44	1,371.2	22.14	1 735.0
800.0	226.9	253.6	338.2	10.32	942.0	14.10	1,282.0	18.40	1 675.2	23.22	2 119.0
900.0	240.7	285.4	380.6	11.00	1,124.0	14.90	1,530.0	19.52	1,998.4	24 75	2 529.0
1,000.0	253.8	316.8	422.4	11.56	1,316.0	15.76	1,791.0	20.64	2,339.2	26.00	2,961.0

TABLE I—(*Continued*)

Head Feet	Velocity Per Sec. Feet	300 M Inch 6 Cu. Ft. H. P.	400 M Inch 8 Cu Ft. H. P.	Diameters of Nozzles						
				5 Inches		5½ Inches		6 Inches		7 Inches
				Cu. Ft.	H. P.	Cu. Ft.	H. P.	Cu. Ft.	H. P.	Cu. Ft.
1.0	8.025	.616	8.8	1.02	.116	1.23	.140	1.49	.100	1.99
1.5	9.83	.948	1.26	1.25	.212	1.51	.257	1.78	.304	2.44
2.0	11.35	1.27	1.69	1.44	.327	1.74	.395	2.08	.464	2.82
2.5	12.68	1.58	2.11	1.61	.457	1.95	.553	2.32	.656	3.15
3.0	13.90	1.90	2.54	1.76	.601	2.13	.727	2.54	.864	3.45
3.5	15.01	2.22	2.96	1.90	.757	2.31	.916	2.74	1.09	3.78
4.0	16.05	2.53	3.37	2.03	.925	2.46	1.12	2.97	1.33	4.09
4.5	17.02	2.84	3.79	2.16	1.10	2.51	1.33	3.10	1.58	4.23
5.0	17.95	3.18	4.24	2.27	1.26	2.75	1.53	3.28	1.81	4.40
6.0	19.66	3.81	5.08	2.49	1.70	3.02	2.05	3.58	2.45	4.88
7.0	21.23	4.44	5.92	2.69	2.14	3 26	2.59	3.87	3.09	5.28
7.5	21.98	4.74	6.32	2.79	2.38	3.42	2.87	4.00	3.42	5.40
10.0	25.38	6.36	8.48	3.22	3.66	3.89	4.42	4.64	5 28	6.30
12.5	28.37	7.92	10.56	3.59	5.11	4.30	6.18	5.20	7.36	7.05
15.0	31.08	9.54	12.72	3.94	6.72	4.76	8.13	5.68	8.08	7.72
17.5	33.57	11.10	14.80	4.26	8.46	5.15	10.24	6.12	12.52	8.34
20.0	35.80	12.66	16.88	4 55	10.34	5.50	12.51	6.56	14.88	8.92
22.5	38.07	14.28	19.04	4.83	12.34	5.84	14.93	6.96	17.76	9.46
25.0	40.13	15.84	21.12	5.09	14.45	6.16	17.49	7.32	20.80	10.15
27.5	42.08	17.40	23.20	5.34	16.67	6.46	20.18	7.68	24.00	10.40
30.0	43.95	18.12	24.16	5.70	19.00	6.90	22.99	8.20	27.36	11.18
32.5	45.75	20 04	26.72	5.80	21.42	7.02	25.92	8.36	30.88	11.37
35.0	47 47	22.14	29.52	6.02	23.94	7.28	28.97	8.68	33 40	11.80
40.0	50.75	25.32	33.76	6.44	29.25	7.78	83.39	9.28	42.08	12.61
45.0	53.83	29.50	38.00	6.82	34.90	8.26	42.23	9.84	50.24	13.38
50.0	56.75	31.68	42.24	7.19	40.87	8.70	49.46	10.36	58.88	14.10
60.0	62.16	38.04	50.72	7.88	53.72	9.54	65.01	11.36	77.44	15.44
70.0	67.14	44.34	59.12	8.51	67.72	10.30	81.95	12.24	97.60	16.09
80.0	71.78	50.74	67.64	9.10	82.76	11.01	100.1	13.12	119.2	17.84
90.0	76.13	57.18	76.24	9.65	98.72	11.58	119.5	13.84	142.1	18.92
100.0	80.25	63.36	84.48	1.017	115.6	12.31	139.9	14.64	166.6	19.94
125.0	89.72	79.26	95.68	11.38	161.6	13.76	195.0	16.32	232.8	22.30
150.0	98.23	95.10	126.8	12.46	212.5	15.08	257.0	17.92	305.9	24.42
175.0	106.1	111.0	148.0	13.46	267.5	15.29	313.7	19.36	385.1	26.39
200.0	113.5	120 8	169.1	14.34	327.0	17.51	395.7	20.64	470.8	28.20
250.0	127.1	158.5	211.4	16.09	457.0	19.47	553.0	23.20	658.0	31.54
300.0	139.0	190.2	253.6	17.62	601.0	21.33	726.9	25.44	865.2	34.54
350.0	150.1	222.5	296.6	19.04	757.2	22.04	916.3	27.36	1,090.4	37.32
400.0	160.5	253.6	338.2	20.35	925 0	24.62	1,179.0	29.28	1,332.0	29.89
450.0	170.2	285.8	381.1	21.59	1,104.0	26.12	1,335.0	31.04	1,590.0	42.31
500.0	179.4	317.1	422.8	22.75	1,293.0	27.54	1,565.0	32.80	1,864.0	44.00
550.0	188.2	349.2	465.6	23.86	1,491.0	28.88	1,805.0	33.60	2,147.0	46.78
600.0	196.0	380.4	507.2	24.93	1,699.0	30.16	2,056.0	35.08	2,446.0	48.86
700.0	212.3	444.0	592.0	27.18	2,142.0	32.88	2,591.0	39.36	3,085.0	53.26
800.0	226.9	507.3	676.4	28.77	2,616.0	34.92	3,166.0	41.28	3,768.0	56.40
900.0	240.7	570.9	761.2	30.52	3,122.0	36.94	3,778.0	44.00	4,496.0	59.83
1000.0	253.8	633.6	844.8	32.17	3,656.0	38.93	4,424.0	46.24	5,264.0	63.06

TABLE I—(*Continued*)

Head Feet	Velocity Per Sec. Feet	500 M Inch 10 Cu. Ft. H. P.	1,000 M Inch 20 Cu. Ft. H. P.	Diameters of Nozzles: 8 Inches Cu. Ft.	8 Inches H. P.	9 Inches Cu. Ft.	9 Inches H. P.	10 Inches Cu. Ft.	10 Inches H. P.	12 Inches Cu. Ft.	12 Inches H. P.
1.0	8.025	1.06	2.12	2.62	.288	3.35	.360	4.07	.46	5.96	[illegible]
1.5	9.83	1.58	3.16	3.20	.544	3.99	.684	4.99	.85	7.12	[illegible]
2.0	11.35	2.11	4.22	3.71	.832	4.68	1.04	5.76	1.30	8.32	[illegible]
2.5	12.68	2.64	5.28	4.08	1.15	5.22	1.48	6.44	1.83	9.28	[illegible]
3.0	13.90	3.17	6.34	4.56	1.54	5.72	1.94	7.05	2.40	10.16	[illegible]
3.5	15.01	3.70	7.40	4.88	1.92	6.16	2.45	7.62	3.03	10.96	[illegible]
4.0	16.05	4.21	8.42	5.20	2.37	6.58	2.99	8.14	3.70	11.88	[illegible]
4.5	17.02	4.74	9.48	5.52	2.81	6.98	3.26	8.64	4.42	12.40	[illegible]
5.0	17.95	5.28	10.6	5.84	3.26	7.38	4.07	9.10	5.05	13.12	[illegible]
6.0	19.66	6.34	12.7	6.40	4.36	8.06	5.51	9.97	6.80	14.32	[illegible]
7.0	21.23	7.39	14.8	6.92	5.52	8.71	6.95	10.77	8.57	15.48	[illegible]
7.5	21.98	7.92	15.8	7.12	6.08	9.00	7.70	11.14	9.50	16.00	[illegible]
10.0	25.38	10.6	21.2	8.64	9.36	10.41	11.88	12.87	14.63	18.56	[illegible]
12.5	28.37	13.2	26.4	9.20	13.84	11.70	16.56	14.39	20.44	20.80	[illegible]
15.0	31.08	15.9	31.8	10.12	17.28	12.78	21.78	15.76	26.87	22.72	[illegible]
17.5	33.57	18.5	37.0	10.88	21.76	13.77	28.17	17.03	33.86	24.48	[illegible]
20.0	35.89	21.1	42.2	11.64	26.56	14.76	33.48	18.20	41.37	26.24	[illegible]
22.5	38.07	23.8	47.6	12.36	31.68	15.66	39.96	19.31	49.37	27.84	[illegible]
25.0	40.13	26.4	52.8	13.04	36.96	16.47	46.80	20.35	57.82	29.28	[illegible]
27.5	42.08	29.0	58.0	13.64	42.72	17.28	54.00	21.34	66.70	30.72	[illegible]
30.0	43.95	30.2	60.4	14.60	48.64	18.45	61.56	22.81	76.01	32.80	[illegible]
32.5	45.75	33.4	66.8	14.84	54.88	18.81	69.48	23.20	85.70	33.44	[illegible]
35.0	47.47	36.9	73.8	15.44	61.28	19.53	77.40	24.08	95.78	34.72	[illegible]
40.0	50.75	42.2	84.4	16.48	74.88	20.88	94.68	25.74	117.0	37.12	[illegible]
45.0	53.83	47.5	95.0	17.44	89.60	22.14	113.0	27.30	139 6	39.36	[illegible]
50.0	56.75	52.8	105.6	18.40	105.0	23.31	128.5	28.78	163.5	41.44	[illegible]
60.0	62.16	63.4	126.8	20.16	137.6	25.56	174.2	31.53	214.9	45.44	[illegible]
70.0	67.14	73.9	147.8	21.68	173.4	27.54	219.6	34.06	270.9	48.96	[illegible]
80.0	71.78	84.6	169.0	23.36	211.8	29.52	268.2	36.41	331.0	52.48	[illegible]
90.0	76.13	95.3	190.6	24.64	252.8	31.14	319.7	38.61	394.9	55.36	[illegible]
100.0	80.25	105.6	211.2	26.08	296.3	32.94	374.8	40.70	462.5	58.56	[illegible]
125.0	89.72	132.1	264.2	29.12	414.0	36.72	523.8	45.51	646.5	65.28	[illegible]
150.0	98.28	158.5	317.0	32.00	554.0	40.32	688.3	49.85	849.8	71.68	[illegible]
175.0	106.1	185.0	370.0	34.56	684.8	43.56	866.5	53.85	1,070.0	77.44	[illegible]
200.0	113.5	211.4	422.8	36.80	878.4	46.44	1,059.0	57.56	1,308.0	82.56	[illegible]
250.0	127.1	264.2	528.4	41.28	1,171.0	52.20	1,481.0	64.36	1,828.0	92.80	[illegible]
300.0	139.0	317.0	634.0	45.12	1,536.0	57.24	1,947.0	70.50	2,403.0	101.76	[illegible]
350.0	150.1	370.8	741.6	48.80	1,949.0	61.56	2,453.0	76.15	3,029.0	109.4	[illegible]
400.0	160.5	422.7	845.4	52.16	2,368.0	65.88	2,997.0	81.41	3,700.0	117.1	[illegible]
450.0	170.2	476.4	952.8	55.36	2,829.0	69.84	3,577.0	86.35	4,415.0	124.2	[illegible]
500.0	179.4	528.4	1,057.0	58.24	3 409.0	73.80	4,194.0	91.02	5,172.0	131.2	[illegible]
550.0	188.2	582.2	1,164.0	61.12	3,821.0	75.60	4,831.0	95.46	5,966.0	134.4	[illegible]
600.0	196.6	634.1	1,268.0	63.84	4,352.0	80.28	5,504.0	99.71	6 798.0	142.7	[illegible]
700.0	212.3	739.8	1,480.0	69.76	5,485.0	88.56	6,191.0	108.7	8 567.0	157.4	[illegible]
800.0	226.9	845.5	1,691.0	73.60	6,701.0	92.88	8,478.0	115.1	10,468.0	165.1	[illegible]
900.0	240.7	951.4	1,903.0	78.08	7,994.0	99.00	10,116.0	122.5	12 489.0	176.0	[illegible]
1,000.0	253.8	1,056.0	2,112.0	82.56	9,357.0	104.00	11,844.0	128.7	14,624.0	185.0	[illegible]

(*b*) Again, find 200, nearest to 201.1 ft., in column headed "head in feet," Table I, opposite which, in column headed 6 in., is found 20.64, which multiplied by 50 gives 1,032, or approximately 1,000 miners' inches, which each nozzle is required to discharge. Hence, each nozzle is to be 6 in. in diameter.

23. Carrying Capacity of Pipes.—In analyzing the flow of water, the total head is divided into three parts; viz., that portion which is due to the velocity; that portion which overcomes the resistance of entry; and that portion which overcomes the resistance within the pipes. In long pipes, the first two parts, compared with the last, are quite small. In Table II, the greatest velocity in any pipe is 13.445 feet per second, due to 4.2 feet, the sum of the first and second portions of the total head, while the third portion is 211.2 feet. The head, or fall, in this table refers to the third part of the total head. Table II has been computed on the assumption that the length of any pipe is not less than 1,000 times its diameter. By "clean pipes" is meant pipes that are smooth and straight. When a pipe is slightly rough, to determine its carrying capacity, multiply the number for clean pipes by .886. When pipes are very rough, the number for clean pipes should be multiplied by .773. When the pipes have any of the following ends—bell-mouthed, square-edged, and square-edged projecting into the reservoir—their coefficients are .900, .836, and .734, respectively.

EXAMPLE 1.—The fall being 52.8 feet per mile, what will be the flow through a pipe 22 inches in diameter, (*a*) in cubic feet, (*b*) in miners' inches?

SOLUTION.—(*a*) In Table II find in first column 52.8 ft., opposite which, in column headed 22 in., will be found 21.06 cu. ft. Ans.

(*b*) This, multiplied by 50, gives 1,053 miners' inches. Ans.

EXAMPLE 2.—The diameter of the pipe being 24 inches, what fall will be required for the pipe to carry 1,000 miners' inches?

SOLUTION.—In Table II, in column headed 24 in., find that number which multiplied by 50 will make 1,000 miners' inches. In this case the nearest number is 20.42, opposite which, in the column Fall, Per Mile, will be found 31.68 ft., the fall required. Ans.

TABLE II

FLOW OF WATER PER SECOND THROUGH CLEAN IRON PIPES

Fall Per Mile	Fall Per Rod		Diameters					
Feet	Ft.	In.	½-Inch Cu. Ft.	¾-Inch Cu. Ft.	1-Inch Cu. Ft.	1¼-Inch Cu Ft.	1½-Inch Cu. Ft.	2-Inch Cu. Ft.
21.12		.792						.02584
26.40		9.990					.02014	.02924
31.68		1.188				.01460	.02270	.03274
36.96		1.396				.01583	.02426	.03492
42.24		1.584			.00567	.01707	.02638	.03776
47.52		1.782			.00617	.01816	.02838	.04081
52.80		1.980		.00316	.00677	.01963	.02988	.04321
63.36		2.376	.00122	.00350	.00781	.02123	.03260	.04843
73.92		2.772	.00124	.00377	.00841	.02282	.03556	.05150
84.48		3.168	.00135	.00411	.00886	.02466	.03706	.05456
95.04		3.564	.00143	.00445	.00961	.02577	.03923	.05740
105.60		3.960	.00150	.00466	.00990	.02793	.04224	.06111
158.40		5.940	.00197	.00589	.01245	.03458	.05175	.07399
211.20		7.920	.00241	.00705	.01492	.04132	.06167	.08734
264.00		9.900	.00279	.00798	.01666	.04577	.07145	.10950
316.80		11.880	.00315	.00874	.01857	.05043	.07830	.12000
369.60	1	1.860	.00340	.00951	.01988	.05424	.08381	.12880
422.40	1	3.840	.00366	.01012	.02141	.05804	.08949	.13750
475.20	1	5.820	.00389	.01086	.02283	.06191	.09400	.14420
528.00	1	7 800	.00410	.01144	.02424	.06724	.10030	.15230
633.00	1	11.760	.00453	.01282	.02676	.07400	.11100	.16340
739.20	2	3.720	.00473	.01380	.02890	.08020	.12000	.17480
844.00	2	7.680	.00524	.01480	.03081	.08622	.12850	.18550
950.40	2	11.640	.00559	.01567	.03276	.09225	.13720	.19550
1,056.00	3	3.600	.00589	.01656	.03458	.09692	.14500	.20470
1,320.00	4	1.500	.00660	.01871	.03897	.10790	.16170	.22760
1,584.00	4	11.400	.00732	.02064	.04316	.11870	.17730	.24830
2,112.00	6	7.200	.00855	.02390	.04987	.13800	.20500	.28330
2,640.00	8	3.000	.00966	.02705	.05648	.15500		
3,168.00	9	10.800	.01065	.03003	.06320			
3,696.00	11	6.600	.01156	.03301	.06943			
4,224.00	13	2.400	.01248	.03572				
4,752.00	14	10.200	.01338	.03786				
5,280.00	16	5.000	.01419					

TABLE II—(*Continued*)

Fall Per Mile	Fall Per Rod		Diameters						
			3-Inch	4-Inch	6-Inch	8-Inch	10-Inch	11-Inch	12-Inch
Feet	Ft.	In.	Cu. Ft.	Cu. Ft.	Cu. Ft.	Cu. Ft.	Cu. Ft.	Cu. Ft.	Cu. Ft.
5.280		.198							1.265
6.336		.238					.878	1.120	1.402
7.392		.277					.960	1.221	1.489
8.448		.317				.573	1.047	1.320	1.634
9.504		.356				.611	1.110	1.394	1.728
10.560		.396			.298	.639	1.194	1.490	1.826
11.616		.436			.314	.659	1.265	1.580	1.940
12.672		.475			.330	.703	1.325	1.653	2.026
13.728		.515		.1235	.346	.737	1.377	1.722	2.117
14.784		.554		.1298	.359	.768	1.423	1.788	2.207
15.840		.594	.0630	.1335	.377	.808	1.470	1.854	2.297
18.480		.684	.0692	.1465	.395	.876	1.587	1.996	2.466
21.120		.792	.0749	.1562	.444	.931	1.683	2.136	2.662
26.400		.990	.0839	.1771	.496	1.045	1.865	2.397	3.020
31.680		1.188	.0915	.1923	.548	1.575	2.059	2.636	3.310
36.960		1.386	.0992	.2146	.589	1.262	2.222	2.858	3.601
42.240		1.584	.1060	.2339	.631	1.344	2.383	3.062	3.856
47.520		1.782	.1119	.2460	.672	1.424	2.514	3.232	4.072
52.800		1.980	.1190	.2582	.721	1.496	2.662	3.419	4.305
63.360		2.376	.1313	.2893	.784	1.644	2.932	3.760	4.728
73.920		2.772	.1413	.3036	.858	1.782	3.210	4.016	5.094
84.480		3.168	.1507	.3237	.922	1.916	3.450	4.390	5.482
95.040		3.564	.1590	.3412	.975	2.033	3.679	4.679	5.839
105.600		3.960	.1717	.3607	1.022	2.155	3.856	5.251	6.160
158.400		5.940	.2081	.4503	1.263	2.667	4.762	6.086	7.630
211.200		7.920	.2469	.5331	1.484	3.145	5.563	7.022	8.860
264.000		9.900	.2785	.5954	1.665	3.513	6.704	8.244	9.967
316.800		11.880	.3049	.6390	1.929	3.847			
369.000	1	1.860	.3331	.6967	1.976	4.196			
422.400	1	3.840	.3559	.7506	2.144				
475.200	1	5.820	.3816	.7960	2.274				
528.000	1	7.800	.4043	.9464	2.399				
633.600	1	11.760	.4440	.9270					
739.200	2	3.720	.4977	1.0060					
844.800	2	7.680	.5131	1.0810					
950.400	2	11.640	.5436						
1,056.000	3	3.600	.5832						
1,320.000	4	1.500	.6523						
1,584.000	4	11.400							

TABLE II—(*Continued*)

Fall Per Mile Feet	Fall Per Rod Inches	Diameters							
		14-Inch Cu. Ft.	15-Inch Cu. Ft.	16-Inch Cu. Ft.	18 Inch Cu Ft.	20-Inch Cu. Ft.	22-Inch Cu. Ft.	24-Inch Cu. Ft.	27-Inch Cu. Ft.
2.11	.08								
2.64	.10								8.27
3.17	.12					3.61	4.61	6.10	8.37
3.70	.14			2.25	3.10	4.07	5.25	6.64	9.09
4.22	.16	1.71	2.05	2.43	3.27	4.35	5.62	7.13	9.48
4.75	.18	1.83	2.19	2.59	3.49	4.68	6.01	7.56	10.26
5.28	.20	1.91	2.30	2.72	3.66	4.92	6.32	7.95	10.74
5.81	.22	2.02	2.43	2.88	3.88	5.15	6.62	8.34	11.45
6.34	.24	2.11	2.54	3.02	4.06	5.40	6.94	8.75	11.93
6.86	.26	2.18	2.65	3.18	4.23	5.62	7.24	9.14	12.54
7.39	.28	2.27	2.75	3.28	4.40	5.82	7.51	9.47	12.96
7.92	.30	2.35	2.84	3.39	4.61	6.05	7.78	9.80	13.49
8.45	.32	2.44	2.94	3.49	4.75	6.27	8.03	10.13	13.98
8.98	.34	2.54	2.98	3.62	4.90	6.48	8.36	10.57	14.41
9.50	.36	2.59	3.11	3.69	5.03	6.65	8.55	10.77	14.81
10.03	.38	2.67	3.21	3.81	5.17	6.92	8.85	11.10	15.21
10.56	.40	2.72	3.29	3.92	5.30	7.05	9.07	11.43	15.63
11.62	.44	2.88	3.47	4.12	5.63	7.42	9.55	12.05	16.44
12.67	.48	3.02	3.63	4.32	5.87	7.79	10.01	12.01	17.23
13.73	.51	3.15	3.79	4.51	6.18	8.14	10.48	13.23	18.01
14.78	.55	3.29	3.95	4.68	6.38	8.48	10.91	13.79	18.75
15.84	.59	3.42	4.11	4.87	6.64	8.77	11.29	14.25	19.50
18.48	.69	3.62	4.46	5.31	7.17	9.49	12.25	15.50	21.13
21.12	.79	3.99	4.78	5.67	7.65	10.16	13.12	16.62	22.62
26.40	.99	4.46	5.37	6.39	8.66	11.43	14.78	18.71	25.34
31.68	1.19	4.91	5.91	7.02	9.54	12.59	16.20	20.42	27.74
36.96	1.39	5.37	6.45	7.66	10.33	13.66	17.53	22.05	29.96
42.24	1.59	5.77	6.90	8.16	11.09	14.66	18.78	23.61	31.99
47.52	1.78	6.11	7.31	8.64	11.71	15.54	19.93	25.07	33.97
52.80	1.98	6.44	7.70	9.10	12.37	16.47	21.06	26.42	35.89
63.36	2.38	7.00	8.39	9.95	13.65	17.99	23.07	29.03	39.76
73.92	2.77	7.60	9.15	10.87	14.75	19.49	24.68	31.49	43.22
84.48	3.17	8.17	9.81	11.63	15.84	21.03	26.97	33.90	46.57
95.04	3.56	8.93	10.47	12.43	16.90	22.45	29.70	36.18	48.06
105.60	3.96	9.26	11.09	13.14	17.85	23.56	31.15	38.45	
158.40	5.94	11.39	13.66	16.17	21.86	28.86			
211.20	7.92	13.22	15.84	18.77					

TABLE II—(*Continued*)

Fall Per Mile Feet	Fall Per Rod Inches	Diameters 30-Inch Cu. Ft.	33-Inch Cu. Ft.	36-Inch Cu. Ft.	40-Inch Cu. Ft.	44-Inch Cu. Ft.	48-Inch Cu. Ft.
1.06	.04			10.29	13.88	18.15	22.98
1.58	.06	7.78	10.21	12.70	17.00	22.22	27.89
2.11	.08	8.99	11.65	14.56	19.68	25.55	32.93
2.64	.10	10.24	12.92	16.35	22.08	28.87	37.00
3.17	.12	10.97	13.99	18.02	24.43	31.46	40.21
3.70	.14	11.90	15.14	19.76	26.27	34.47	43.67
4.22	.16	12.84	16.36	20.85	28.14	37.05	46.81
4.75	.18	13.48	17.58	22.30	29.80	39 01	49.06
5.28	.20	14.21	18.74	23.47	31.46	41.06	52.15
5.81	.22	15.05	19.54	24.91	33.25	42.09	54.95
6.34	.24	15.81	20.28	26.12	34.68	44.97	57.36
6.86	.26	16.47	21.29	27.20	36.21	46.77	60.07
7.39	.28	17.18	22.20	28.24	37.57	48.83	62.02
7.92	.30	17.94	23.01	29.19	39.18	50.62	64.47
8.45	.32	18.58	23.76	30.29	40.54	52.46	66.53
8.98	.34	19.21	24.47	31.42	41.88	54.04	68.50
9.50	.36	19.66	25.22	32.48	43.07	55.48	70.62
10.03	.38	20.32	26.14	33.40	44.28	57.01	72.75
10.56	.40	20.79	26.94	34.49	45.20	58.85	74.44
11.62	.44	21.80	28.27	36.15	48.12	61.71	78.29
12.67	.48	22.83	29.02	37.74	50.48	64.35	81.68
13.73	.51	23.93	31.06	39.40	52.67	66.87	85.20
14.78	.55	24.86	32.28	40.86	55.04	69.57	88.46
15.84	.59	25.87	33.62	42.28	56.33	72.32	91.73
18.48	.69	27.96	36.17	45.95	61.09	77.95	100.40
21.12	.79	29.84	38.57	48.83	65.41	83.60	105.89
26.40	.99	33.55	43.12	54.89	73.09	93.37	119.34
31.68	1.19	36.79	47.40	59.95	80.32	103.28	130.88
36.96	1.39	39.66	51.35	65.17	86.70	111.74	148.09
42.24	1.59	42.39	54.91	69.80	92.58	119.93	153.94
47.52	1.78	45.23	58.20	74.33	98.00	128.26	
52.80	1.98	47.71	61.62	78.46	103.99		
63.36	2.38	52.91	68.00	82.84			
73.92	2.77	57.65	73.95				

TABLE II—(*Continued*)

Fall Per Mile Feet	Fall Per Rod Inches	Diameters 54-Inch Cu. Ft.	60-Inch Cu. Ft.	72-Inch Cu. Ft.	84-Inch Cu. Ft.	96-Inch Cu. Ft.
.53	.02	21.96	29.77	46.99	75.43	107.77
1.06	.04	31.70	38.19	57.65	104.61	152.45
1.58	.06	38.53	52.09	82.53	126.18	188.45
2.11	.08	45.12	59.04	95.99	145.43	218.75
2.64	.10	50.23	67.56	109.42	162.75	245.30
3.17	.12	55.51	74.32	121.58	177.03	267.41
3.70	.14	60.21	80.51	132.04	192.04	290.53
4.22	.16	63.61	86.30	139.96	207.81	310.89
4.75	.18	67.20	91.99	148.72	222.44	324.20
5.28	.20	72.37	96.98	157.77	235.13	350.45
5.81	.22	75.71	102.39	165.97	253.34	366.19
6.34	.24	79.13	107.31	173.04	264.77	382.02
6.86	.26	82.54	115.53	179.26	275.16	397.85
7.39	.28	85.90	116.53	187.46	287.67	414.70
7.92	.30	89.52	119.68	193.93	296.37	427.76
8.45	.32	92.43	123.70	200.18	307.87	443.09
8.98	.34	95.35	127.63	206.40	316.15	457.42
9.50	.36	97.65	131.26	212.05	326.73	470.49
10.03	.38	100.19	134.79	217.71	335.79	481.53
10.56	.40	103.82	138.84	225.21	348.25	496.37
11.62	.44	108.78	145.98	235.52	364.92	522.76
12.67	.48	113.47	152.56	246.41	389.09	547.88
13.73	.51	118.48	158.65	256.17	394.43	510.01
14.78	.55	123.10	164.54	267.19	408.36	592.13
15.84	.59	128.19	170.43	277.88	423.36	612.00
18.48	.69	138.92	183.98	299.72	482.99	
21.12	.79	147.91	197.52	320.74		
26.40	.99	165.80	221.95	358.52		
31.68	1.19	182.42	244.26			
36.96	1.39	190.01				

EXAMPLE 3.—In carrying 1,050 inches of water to a hydraulic mine in a pipe 27 inches diameter, having a fall of 95.04 feet to the mile, what will be the effective head at the mine?

SOLUTION.—In Table II, in column headed 27 in., find that number which multiplied by 50 will make 1,050 approximate miners' inches.

TABLE III

ADDITIONAL HEAD REQUIRED TO OVERCOME THE RESISTANCE OF ONE CIRCULAR BEND

Velocity Per Second Feet	Ratio of Radius of Pipe to Radius of Bend									
	1 : 5 30° Head Feet	1 : 5 60° Head Feet	1 : 5 90° Head Feet	1 : 5 120° Head Feet	1 : 5 180° Head Feet	2 : 5 30° Head Feet	2 : 5 60° Head Feet	2 : 5 90° Head Feet	2 : 5 120° Head Feet	2 : 5 180° Head Feet
1	.0004	.0007	.0011	.0014	.0022	.0005.	.001	.002	.002	.005
2	.0014	.0029	.0043	.0058	.0086	.0021	.004	.006	.008	.013
3	.0032	.0064	.0096	.0128	.0192	.0048	.010	.014	.020	.029
4	.0057	.0114	.0171	.0228	.0342	.0085	.017	.025	.034	.051
5	.0089	.0179	.0268	.0358	.0536	.0133	.027	.040	.054	.080
6	.0129	.0257	.0386	.0514	.0772	.0192	.038	.058.	.076	.115
7	.0175	.0350	.0525	.0700	.1050	.0261	.052	.078	.104	.157
8	.0229	.0457	.0686	.0914	.1372	.0341	.068	.102	.136	.205
10	.0357	.0714	.1072	.1428	.2144	.0533	.107	.160	.214	.32
15	.0804	.1607	.2411	.3214	.4822	.1200	.240	.360	.480	.72
20	.1429	.2858	.4287	.5716	.8574	.2130	.426	.639	.852	1.28
25	.2232	.4464	.6696	.8928	1.34	.3333	.667	1.00	1.33	2.00
30	.3214	.6428	.9642	1.29	1.93	.4798	.96	1.44	1.92	2.88
40	.5714	1.14	1.71	2.28	3.42	.853	1.71	2.56	3.42	5.12
50	.8927	1.79	2.68	3.58	5.36	1.33	2.66	3.99	5.32	7.98
75	2.01	4.02	6.03	8.04	12.06	3.00	6.00	9.00	12.00	18.00
100	3.57	7.14	10.71	14.28	21.42	5.33	10.67	15.99	21.34	31.98
150	8.04	16.07	24.11	32.14	48.22	12.00	24.00	36.00	48.00	72.00
200	14.29	28.58	42.87	57.16	85.74	21.32	42.64	63.96	127.92	127.92
300	32.14	64.28	96.42	128.56	192.84	47.98	95.96	143.94	287.88	287.88

In this case we have 21.13 cu. ft., opposite which, in the column Fall Per Mile, we find 18.48 ft., which is the head per mile lost in carrying the water. Subtracting this from the given fall, or head, gives the effective head. Thus, 95.04 − 18.48 = 76.56 ft. effective head. Ans.

EXAMPLE 4.—There being 7.5 gallons in a cubic foot, and 86,400 seconds in a day (24 hours), the fall 7.39 feet per mile, how many gallons will a pipe 40 inches diameter carry per day?

SOLUTION.—In Table II, in column headed 40 in., and opposite 7.39 ft., in column Fall Per Mile, will be found 37.57 cu. ft. flow per sec. Then, 37.57 × 7.5 × 86,400 = 24,345,360 gal. Ans.

As a general rule the velocity per second is equal to 50 times the square root of the product of the head and diameter in feet, divided by the sum of the length and 50 times the diameter of the pipe in feet. This rule applies to both long and short pipes, and is approximately accurate if the diameter does not exceed 2 feet.

24. Resistance of Bends to Flow of Water.—Tables III and IV show the resistance to the flow of water that is offered by both circular and angular bends. Their use is illustrated by the following examples:

EXAMPLE 1.—The radius of the pipe being to the radius of the bend in the ratio of 1 : 5, the number of degrees in the bend being 90, and the velocity 75 feet per second, what is the additional head required to overcome the resistance of the bend?

SOLUTION.—In Table III, in the column Velocity Per Second, find 75 ft., opposite which, in column headed 1 : 5, 90°, is found 6.03 ft., the required head. Ans.

EXAMPLE 2.—The radius of the pipe being to the radius of the bend in the ratio of 2 : 5, the number of degrees in the bend being 120, and the velocity per second 100 feet, what is the additional head required to overcome the resistance of one bend?

SOLUTION.—In Table III, opposite 100 ft. velocity, will be found, in column headed 2 : 5, 120°, the required number, viz., 21.34 ft. Ans.

EXAMPLE 3.—The velocity being 40 feet per second, what additional head is required to overcome the resistance of an angular bend whose angle of deflection is 90°?

SOLUTION.—In Table IV find in the column Velocity Per Second, 40, opposite which, in column headed 90° head, will be found 24.45 ft., the additional head required. Ans.

TABLE IV

ADDITIONAL HEAD REQUIRED TO OVERCOME THE RESISTANCE OF ONE ANGULAR BEND

Velocity Per Second Feet	Angles of Deflection					
	15° Head Feet	30° Head Feet	40° Head Feet	60° Head Feet	90° Head Feet	120° Head Feet
1	.0002	.0005	.002	.006	.015	.029
2	.0010	.0019	.009	.023	.061	.116
3	.0022	.0042	.019	.051	.138	.260
4	.004	.008	.035	.090	.245	.462
5	.006	.012	.054	.141	.382	.723
6	.009	.017	.078	.204	.550	1.04
7	.012	.023	.106	.277	.749	1.42
8	.016	.030	.138	.362	.978	1.85
10	.025	.047	.216	.565	1.53	2.89
15	.056	.105	.486	1.27	3.44	6.50
20	.099	.186	.863	2.26	4.85	11.56
25	.155	.291	1.35	4.45	9.55	18.06
30	.224	.419	1.94	5.09	13.75	26.01
40	.398	.745	3.45	9.04	24.45	46.23
50	.621	1.17	5.40	14.13	38.20	73.93
75	1.40	2.62	12.14	31.79	85.95	162.5
100	2.48	4.66	21.58	56.52	152.8	289.0
150	5.59	10.48	48.57	127.2	343.7	650.2
200	9.94	18.63	86.32	226.1	611.1	1,156.
300	22.36	41.92	194.20	508.7	1,092.	2,601.

25. Loss of Head by Friction.—Table V shows the loss of head by friction in each 100 feet in length of different diameters of pipe when discharging the given quantities of water per minute.

26. Calculating the Horsepower of Water.—The horsepower given in Tables VI and VII equals 85 per cent. of the theoretical horsepower. Table VI gives the horsepower of 1 miners' inch of water under heads from 1 to 1,100 feet. This inch equals 1½ cubic feet per minute. Table VII gives the horsepower of 1 cubic foot of water per

TABLE V

LOSS OF HEAD IN PIPE BY FRICTION

Velocity in Feet Per Second	Inside Diameter of Pipe in Inches: 13		14		15		16		18		20	
	Loss of Head in Feet	Cubic Feet Per Minute	Loss of Head in Feet	Cubic Feet Per Minute	Loss of Head in Feet	Cubic Feet Per Minute	Loss of Head in Feet	Cubic Feet Per Minute	Loss of Head in Feet	Cubic Feet Per Minute	Loss of Head in Feet	Cubic Feet Per Minute
2.0	.183	110	.169	128	.158	147	.147	167	.132	212	.119	262
2.2	.216	121	.200	141	.187	162	.175	184	.156	233	.140	288
2.4	.252	133	.234	154	.218	176	.205	201	.182	254	.164	314
2.6	.290	144	.270	167	.252	191	.236	218	.210	275	.189	340
2.8	.332	156	.308	179	.288	206	.270	234	.240	297	.216	366
3.0	.375	166	.349	192	.325	221	.306	251	.271	318	.245	393
3.2	.422	177	.392	205	.366	235	.343	268	.305	339	.275	419
3.4	.471	188	.438	218	.408	250	.383	284	.339	360	.306	445
3.6	.522	199	.485	231	.452	265	.425	301	.377	382	.339	471
3.8	.576	210	.535	243	.499	280	.468	318	.416	403	.374	497
4.0	.632	221	.587	256	.548	294	.513	335	.456	424	.410	523
4.2	.691	232	.641	269	.598	309	.561	352	.499	445	.449	550
4.4	.751	243	.698	282	.651	324	.611	368	.542	466	.488	576
4.6	.815	254	.757	295	.707	339	.662	385	.588	488	.529	602
4.8	.881	265	.818	308	.763	353	.715	402	.636	509	.572	628
5.0	.949	276	.881	321	.822	368	.770	419	.685	530	.617	654
5.2	1.020	287	.947	333	.883	383	.828	435	.736	551	.662	680
5.4	1.092	298	1.014	346	.947	397	.888	452	.788	572	.710	707
5.6	1.167	309	1.083	359	1.011	412	.949	469	.843	594	.758	733
5.8	1.245	321	1.155	372	1.078	427	1.011	486	.899	615	.809	759
6.0	1.325	332	1.229	385	1.148	442	1.076	502	.957	636	.861	785
7.0	1.750	387	1.630	449	1.520	515	1.430	586	1.270	742	1.143	916

TABLE V—(*Continued*)

Velocity in Feet Per Second	22: Loss of Head in Feet	22: Cubic Feet Per Minute	24: Loss of Head in Feet	24: Cubic Feet Per Minute	26: Loss of Head in Feet	26: Cubic Feet Per Minute	28: Loss of Head in Feet	28: Cubic Feet Per Minute	30: Loss of Head in Feet	30: Cubic Feet Per Minute	36: Loss of Head in Feet	36: Cubic Feet Per Minute
	Inside Diameter of Pipe in Inches											
2.0	.108	316	.098	377	.091	442	.084	513	.079	589	.066	848
2.2	.127	348	.116	414	.108	486	.099	564	.093	648	.078	933
2.4	.149	380	.136	452	.126	531	.116	616	.109	707	.091	1,018
2.6	.171	412	.157	490	.145	575	.134	667	.126	766	.104	1,100
2.8	.195	443	.180	528	.165	619	.153	718	.144	824	.119	1,188
3.0	.222	475	.204	565	.188	663	.174	770	.163	883	.135	1,273
3.2	.249	507	.220	603	.211	708	.195	821	.182	942	.152	1,357
3.4	.278	538	.255	641	.235	752	.218	872	.204	1,001	.169	1,442
3.6	.308	570	.283	678	.261	796	.242	923	.226	1,060	.188	1,527
3.8	.340	601	.312	716	.288	840	.267	974	.249	1,119	.207	1,612
4.0	.373	633	.342	754	.315	885	.293	1,026	.273	1,178	.228	1,697
4.2	.408	665	.374	791	.345	929	.320	1,077	.299	1,237	.249	1,782
4.4	.444	697	.407	829	.375	973	.348	1,129	.325	1,296	.271	1,866
4.6	.482	728	.441	867	.407	1,017	.378	1,180	.353	1,355	.294	1,951
4.8	.521	760	.476	905	.440	1,062	.409	1,231	.381	1,414	.318	2,036
5.0	.561	792	.513	942	.474	1,106	.440	1,283	.411	1,472	.342	2,121
5.2	.602	823	.552	980	.510	1,150	.473	1,334	.441	1,531	.368	2,206
5.4	.645	855	.591	1,018	.546	1,194	.507	1,385	.473	1,590	.394	2,291
5.6	.690	887	.632	1,055	.583	1,239	.542	1,437	.506	1,649	.421	2,376
5.8	.735	918	.674	1,093	.622	1,283	.578	1,488	.540	1,708	.450	2,460
6.0	.782	950	.717	1,131	.662	1,327	.615	1,539	.574	1,767	.479	2,545
7.0	1.040	1,109	.953	1,319	.879	1,548	.817	1,796	.762	2,061	.636	2,868

TABLE VI

MINERS'-INCH TABLE

Head in Feet	Horsepower	Head in Feet	Horsepower
1	.0024147	320	.772704
20	.0482294	330	.796851
30	.072441	340	.820998
40	.096588	350	.845145
50	.120735	360	.869292
60	.144882	370	.893439
70	.169029	380	.917586
80	.193176	390	.941733
90	.217323	400	.965880
100	.241470	410	.990027
110	.265617	420	1.014174
120	.289764	430	1.038321
130	.313911	440	1.062468
140	.338058	450	1.086615
150	.362205	460	1.110762
160	.386352	470	1.134909
170	.410499	480	1.159056
180	.434646	490	1.183206
190	.458793	500	1.207350
200	.482940	520	1.255644
210	.507087	540	1.303938
220	.531234	560	1.352232
230	.555381	580	1.400526
240	.579528	600	1.448820
250	.603675	650	1.569555
260	.627822	700	1.690290
270	.651969	750	1.811025
280	.676116	800	1.931760
290	.700263	900	2.173230
300	.724410	1,000	2.414700
310	.748557	1,100	2.656170

TABLE VII

CUBIC-FOOT TABLE

Head in Feet	Horsepower	Head in Feet	Horsepower
1	.0016098	320	.515136
20	.032196	330	.531234
30	.048294	340	.547332
40	.064392	350	.563430
50	.080490	360	.579528
60	.096588	370	.595626
70	.112686	380	.611724
80	.128784	390	.627822
90	.144892	400	.643920
100	.160980	410	.660018
110	.177078	420	.676116
120	.193176	430	.692214
130	.209274	440	.708312
140	.225372	450	.724410
150	.241470	460	.740508
160	.257568	470	.756606
170	.273666	480	.772704
180	.289764	490	.788802
190	.305862	500	.804900
200	.321960	520	.837096
210	.338058	540	.869292
220	.354156	560	.901488
230	.370254	580	.933684
240	.386352	600	.965880
250	.402450	650	1.046370
260	.418548	700	1.126860
270	.434646	750	1.207350
280	.450744	800	1.287840
290	.466842	900	1.448820
300	.482940	1,000	1.609800
310	.499038	1,100	1.770780

minute under heads of from 1 to 1,100 feet. The following example illustrates the use of Table VI when the exact number of feet head is given.

When the exact head is not found in Table VI, take the horsepower of 1 inch under 1 foot head and multiply by the number of inches, and then by the number of feet head. The product will be the required horsepower.

This rule may be used for the cubic feet in Table VII, by substituting the equivalents therein for those of miners' inches.

EXAMPLE.—Having 100 feet head and 50 miners' inches of water, what is the horsepower?

SOLUTION.—Under Head in Feet find 100, and opposite in the second column .241470 will be seen. Multiply this by 50, then $.24147 \times 50 = 12.07$ H. P. Ans.

DISPOSITION OF PLACER DEBRIS

27. Placer Tailings.—In large placer mines the disposition of the debris has always been a serious question. In many cases it is washed into some rapidly flowing river, but sooner or later it is liable to work down the stream and block the course of the river, causing much damage to agricultural land or serious floods. On this account, a permanent injunction has been granted against certain classes of placer mining in California. To overcome the difficulties complained of and to dispose of debris several devices have been adopted.

Dams of brush or logs are sometimes constructed in cañons and the debris allowed to accumulate behind them. As the level of the debris reaches that of the top of the dam, the height of the dam is increased. By carefully constructing the dams and by keeping them to a height sufficient to hold back the debris, the latter may be retained in the gorge; but in some cases the level of the debris would soon reach that of the ground from which it was excavated, and hence work would be brought to a standstill. To avoid this an elevator for raising the debris was adopted. In

some cases the debris has been elevated mechanically by means of bucket elevators, but as this system is expensive it is now rarely adopted, hydraulic elevators having replaced them.

Sometimes dams or obstructions made of stone, timber, or brush are placed across the beds of streams to hold back the waste from mines and to prevent damage in the valleys below. The difference between a debris dam and a water dam is that no attempt is made to render the debris dam water-tight, the only object being that it will retard the

FIG. 11

flow of the stream and give it a greater breadth of discharge, so that the stream will naturally have to deposit the sediment that it is carrying. The sediment silts, or fills up, against the upper face of the dam, so that the area above the dam soon becomes a flat expanse over which the water finds its way to the dam. When these dams are constructed of stone, the individual stones in the lower face

and crest of the dam should be so large that the current will be unable to displace them, while the upper face and core of the dam may be composed of finer material. In case a break should occur in a debris dam, it will not necessarily endanger the region farther down the stream, as is the case where a break occurs in a water dam. The reason for this is that the debris dam is not made water-tight, and hence there is never much pressure against it. In case a breach should occur, the only result would be that more or less of the gravel held behind the dam would be washed through the breach and down the stream. Fig. 11 shows the debris-restraining dam below the Red Dog Hydraulic Mine, Nevada County, California. The method of constructing the log cribs for such dams is shown.

28. The Ludlum Hydraulic Elevator.—One of the first hydraulic elevators that was constructed to raise tailings is illustrated in Fig. 12; the principle is that of the ejector. A stream of water under considerable pressure is discharged through a nozzle, as shown in the illustration,

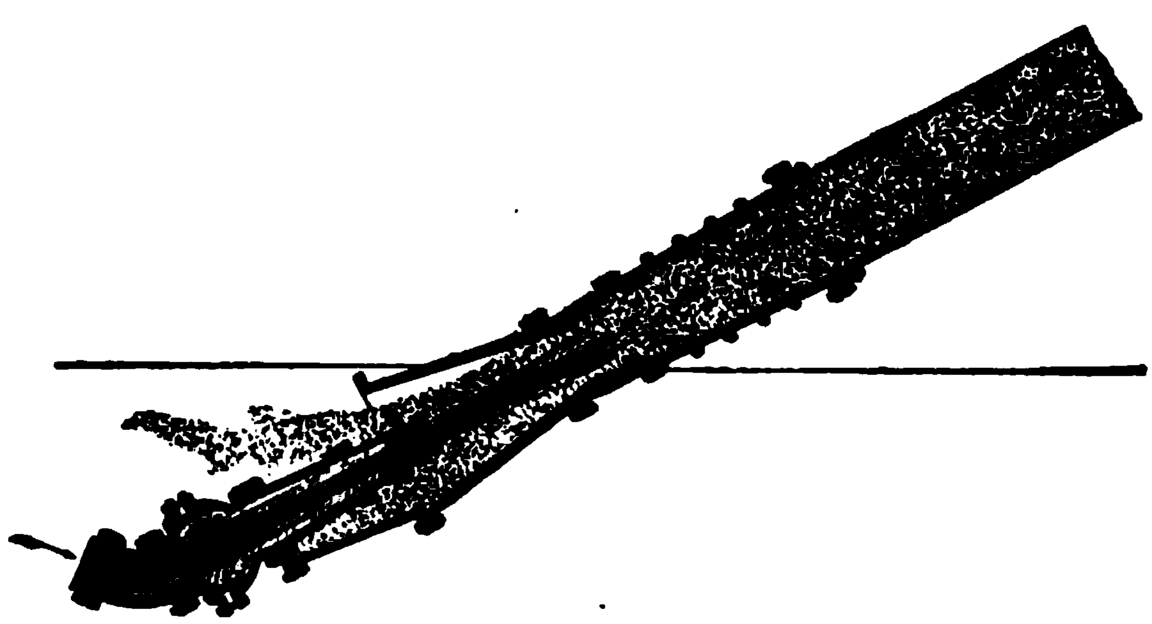


FIG. 12

and gravel mixed with water is fed to the opening around the nozzle. The force of the stream from the nozzle carries the entire mass up the discharge pipe to a considerable height. Similar elevators are also constructed for removing water from river beds or other places that have a tendency to become flooded.

29. Evans Elevator.—Fig. 13 illustrates the **Evans elevator,** which works on the same principle as the ejector. The water under pressure enters through *a* and is discharged through a nozzle *d*. The suction for the gravel is through a large opening in front. The rapid flow of the water draws the material in and ejects it through the contracted portion *e* and on through the discharge pipe. One advantage claimed for this elevator is the introduction of the two auxiliary suctions *b* and *c*. These are placed at the sides of the main suction and may be employed for draining the pit of water, for drawing in fine material, or even for draining places in bed rock at some distance from the elevator. One advantage in the use of these auxiliary suctions is that the space back of the nozzle is thus provided with a flow of water, and the boulders and other material drawn in through the main suction prevented from rushing around into this space with a great force, which they would otherwise have a tendency to do. These elevators are built with a capacity to handle stones 18 inches in diameter and lift material 15 feet high for every 100 feet head of water.

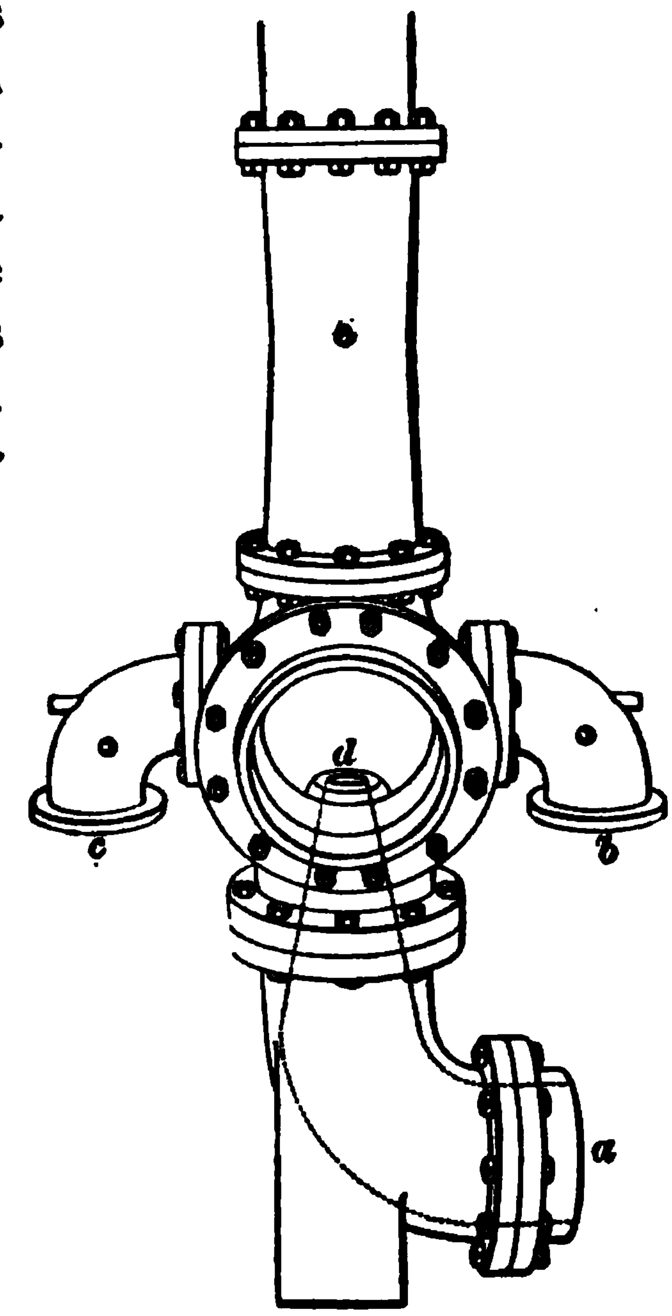


FIG. 13

AUXILIARY APPARATUS

30. Strong **derricks** are used in hydraulic mining to remove heavy boulders. The mast is frequently 100 feet high and the boom over 90 feet long. The mast is held in position by guy wires and is provided with the proper tackle

both for raising and lowering the boom and for raising weights attached to the boom. Such derricks are usually operated by means of waterwheels of the impulse pattern,

FIG. 14

shown in Fig. 14. The wheels are sometimes 10 feet in diameter and are connected to the drums that operate the ropes by means of gearing.

31. Waterwheels.—The "hurdy-gurdy" or impact wheel is moved by a jet of water issuing from a nozzle under pressure; the water strikes against buckets on the circumference of the wheel. One form, known as the Pelton waterwheel, is shown in Fig. 14. Undershot waterwheels are also employed to operate the Chinese pumps or bucket elevators in connection with wing dams in bar mining.

32. Lighting.—Where mines are worked night and day some form of artificial light must be employed for the night work. The lights may be locomotive headlights, Wells' lights, or bonfires, but most of the large mines now use electric arc lights.

CLEANING UP

33. The length of the run depends on the wear of the pavement and riffles, and on the value of the material being put through the sluices. Some claims are cleaned up every 20 days, others every 2 or 3 months, and a few only once a season. All pavements should be cleaned as soon as they begin to wear in grooves. Where large quantities of gravel are washed, it is advisable to clean the first 1,000 or 2,000 feet of sluice about every 2 weeks, the tailing sluices being cleaned only once a year. Undercurrents should be cleaned up whenever quicksilver is found spread over the lower riffles with a tendency to discharge over the end of the sluice. The gold-saving tables are cleaned up every few days, without stopping the work in the main sluice, by simply shutting off the water from one table while it is being cleaned up, and repeating the operation with all the others. When it is decided to make a general clean up, the bed rock and ground sluices are washed clean, clear water alone being turned into the sluices until they are free from dirt. A small quantity of water in which a man can conveniently work is then allowed to flow through the sluice, the pavement blocks are pried out with crowbars, washed clean from amalgam, and laid alongside the sluice. This is done in sections of 100 feet or so. One row of blocks is left in the sluice between each section. These rows serve as riffles to prevent the gold and quicksilver from passing down the sluice. After the first section of blocks is taken up, the men follow the gravel and dirt remaining in the sluice as it is slowly washed down and pick out the quicksilver and amalgam with iron spoons. As each riffle is reached the amalgam and quicksilver are collected and placed in sheet-iron buckets, after which the blocks left for riffles are removed and the residue washed down to the next riffle. This is continued the entire length of the sluice. When this operation is finished the water is turned off entirely and workmen go over the sluice with small silver spoons, digging the amalgam out of nail holes and cracks. After this the side lagging is overhauled and

the pavement blocks are replaced. Very long sluices are usually lined in the lower portion with heavy rock riffles, which can be used for a longer period without cleaning up than the smaller riffles used farther up. In some cases it is customary, where mines are run night and day, to replace the lining, and resume washing at night, proceeding thus until the entire sluice is cleaned up.

AMALGAMATION

34. Saving Fine Gold.—Although heavy gold may be arrested by the various contrivances described, such as riffles and undercurrents, much fine gold might escape in the absence of mercury or quicksilver. Mercury will instantly seize and amalgamate any fine gold coming in contact with it. When using zigzag riffles, a vessel containing quicksilver and pierced by a small hole that allows the metal to escape drop by drop, is sometimes placed at the head of the sluice. The quicksilver moving from riffle to riffle overtakes, absorbs, and retains the fine gold, the amalgam thus formed being caught in the regular riffles farther down. Where the sluices are provided with longitudinal riffles after starting the washing, mercury is poured into the head of the sluice and finds its way down with the current, though the larger proportion of it will remain in the upper boxes. Small quantities of quicksilver are sometimes introduced at intervals along the sluice, the quantity being increased in direct proportion to the amount of fine gold present.

When wooden-block riffles are employed, an attempt is sometimes made to impregnate the pores of the wood with mercury. This is accomplished by grinding the end of a piece of gas pipe to a thin edge and driving it into the wood. The gas pipe is then filled with mercury, and the pressure of the column will force a certain amount of the fluid into the pores of the wood. As the wood gradually wears away, this mercury becomes exposed and amalgamates any

gold that may come in contact with it. When cleaning up the sluice, the amalgam remaining on the surface of the wood is simply scraped off.

35. Copper-Plate Amalgamation.— Where gold is very fine, amalgamated copper plates are sometimes employed. These are usually at least 3 feet wide by 6 feet long, while sometimes the stream is split and carried over two or three such plates, so as to reduce its speed as much as possible. The plates are placed nearly level and at a considerable distance from the head of the sluice, as it is intended to catch only the fine gold; for this reason also a screen is placed above the plates in such a manner as to remove all the coarse material and allow only the fine material to pass over them. The screens employed for this purpose are frequently perforated with holes $\frac{1}{8}$ inch by $\frac{1}{16}$ inch, similar to the slotted screens used in the stamp batteries of gold and silver mills. The copper plate is amalgamated by first cleaning its upper surface with dilute nitric acid, and then applying some mercury that has been treated with dilute nitric acid to form a little nitrate of mercury. The current must be slow and shallow, so that every particle of gold may come in contact with the surface of the plate. A freshly amalgamated plate may become coated with a green slime of subsalts of copper. This must be carefully scraped off and the plate carefully rubbed with fresh mercury. To remove the amalgam from the plates, a whisk broom, the bristles of which have been cut short, is used, or the plates may be cleaned with a knife or a rubber scraper made like a hoe. If it is desired to remove practically all the amalgam, this may be done by gently heating the plate and then scraping; but it is usually undesirable to do this, for the plates always catch gold better when there is a little amalgam remaining on them, and warming the plate makes them more difficult to recoat. The copper plates should not be less than $\frac{1}{16}$ inch thick.

36. Amalgam kettles are ordinary sheet-iron buckets or porcelain kettles. They are used as receptacles in which

to collect mercury and amalgam while cleaning up sluices and undercurrents, and also for floating amalgam to free it from foreign substances before it is strained and retorted. No receptacle made from material with which mercury will alloy should be used for this purpose.

37. Cleaning the Amalgam.—The quicksilver and amalgam obtained in cleaning up the sluice and undercurrents are stirred in buckets, if necessary with the addition of mercury to make a fairly fluid mass; the coarse sand, nails, and other foreign substances that float to the surface are then skimmed off. This residue of sand, which always retains some amalgam, is concentrated by working in pans or rockers, and the concentrates are ground in iron mortars with some clean quicksilver to free the amalgam. Any base material floating on the surface after this second cleaning is melted separately to a base bullion, and clean quicksilver and amalgam are added to the clean portion from the first. The quicksilver is then separated from the amalgam by straining through canvas, chamois skin, or buckskin, and the dry amalgam is treated in iron retorts. A barrel into which pieces of iron and quicksilver are placed and revolved, or tumbled, for a few hours may be used to advantage in freeing sand from amalgam. The surplus quicksilver is separated from the amalgam as completely as can be done by straining through buckskin.

Fig. 16

38. Retorting Bullion.—When the quantity of amalgam to be treated is small, the retort shown in Fig. 15 may

be used for volatilizing the mercury in the amalgam. This retort is a cast-iron pot with a heavy flange *a* ground to make a tight joint with the cover *b*. The cover and pot are held in place by the clamp *c* and the screw *d*. In the top of the cover is screwed the bent pipe *e*, which is coupled to a straight pipe *f* surrounded by a cooler *g*. The cooler is for condensing the fumes of mercury that come from the retort when that is placed in a fire. Water flows into the cooler, at *h*, from a reservoir placed higher than the retort and flows out through *i*. The condensed mercury flows from the pipe *f* into a receptacle prepared to receive it, which may also be partially cooled by water.

Where large amounts of amalgam are handled, furnaces with stationary cast-iron retorts are constructed especially for recovering the mercury from the amalgam. If the retort were to be left unattended for a short time, and it were placed immediately above the fire, it might become overheated, and the weight of the metal inside of the retort cause it to "belly," thus ruining it completely. To prevent this, the retort should be supported at several points and arranged with the fire on one side, so that the heat may be evenly distributed over it.

Fig. 16 illustrates this retort and furnace in section. Before putting amalgam into it the inside of any retort

FIG. 16

should be coated with a thin sheet of clay to prevent the amalgam adhering to the iron of the retort. The amalgam

should be carefully introduced and spread out evenly. The pipe connecting the retort with the condenser must be cleaned of all obstructions and the amalgam so spread that in no possible way can the pipe become choked, for if this happens an explosion will probably result; such an explosion will fill the retorting room with the poisonous fumes of mercury and endanger the lives of the workmen. When the amalgam is first placed in the retort, care should be taken that none of it is too close to the pipe connecting with the condenser, for when retorting begins the amalgam swells into a spongy mass and may close the condensing pipe; the cover is luted on with clay or a mixture of clay and wood ashes, and then securely clamped. To avoid the danger of amalgam excessively swelling, the heat should be applied very slowly at first and gradually raised until a dark-red heat is attained, that temperature being all that is necessary to volatilize the quicksilver. Toward the end of the operation, however, the heat may without damage be raised to a cherry red until distillation ceases. During the operation, the condenser coil at the back of the retort should be kept cool by means of a continuous supply of fresh water, which enters from the lower end of the box containing the condenser, while the warm water discharges from the upper end of the box. For the purpose of cleaning and inspection, a straight-tube condenser is preferable to a coil, for it is impossible to run a rod through a coil for the purpose of cleaning; it is also impossible to satisfactorily inspect the coil and see that it is clean. The retorted bullion is cut or broken into pieces and then melted in well-annealed black-lead crucibles, nitric acid being afterwards used to remove the base metals, and the gold then cast into bars.

DREDGING

RIVER DREDGING

DREDGES

39. General Considerations.—Where streams cannot be diverted from their courses by flumes and wing dams so as to expose the river bed, it becomes necessary, in case the deposit is to be worked, to introduce the method of excavating and handling the material termed **dredging.** The **dredge** for working river beds consists of a flat-bottomed scow provided with suitable machinery. The first dredge was a crude affair; it consisted of a small scow provided with a spoon that was thrown out over the stern and allowed to sink to the river bed. A small hand windlass on the scow drew the spoon along the river bottom and then into the scow, where it was dumped and the gravel washed from the gold. This proved so successful that the scows were afterwards built with undershot waterwheels, having an axle to act as a drum. The current of the stream revolved these waterwheels and wound up a rope attached to a scoop that dredged the river beds; this arrangement was found to be so economical and serviceable that its use has been continued in some localities to the present day. This beginning led to great improvements until large scows supplied with modern excavating and concentrating machinery were constructed. Dredges able to excavate low-grade or high-grade river beds and bars, at a moderate working cost, have been adopted in nearly every civilized country where gold dredging can be carried on.

40. Types of Dredges.—There are four recognized types of dredges; they differ from one another chiefly in

their manner of excavating. Whatever may be the type, they are all provided with powerful machinery capable of dealing with large quantities of material. In some localities where electricity is cheaper than steam, dredges are operated by electrical machinery, but as a usual thing steam is the motive power. The concentrating machinery of the different types of dredges differs in construction and application, but is not a distinctive type, similar machinery being adapted to other concentrating purposes.

41. Specifications for Dredges.—The machinery for excavating the ground and conveying it to the concentrators must be strong enough to withstand the great strains that at times such machinery necessarily receives, especially where large boulders or rock ledges are encountered. There must be a serviceable washing apparatus for cleaning the coarse material and separating the gold from adhering dirt; also gold-saving devices, such as tables, sluices, and riffles, in order to collect whatever gold is excavated. There must be some kind of machinery for discharging the tailings in such a manner that they will not interfere with the present or future dredging and the movements of the scow. Arrangements must be provided for moving the scow from place to place, as the material is excavated to bed rock, and also for holding the scow up to the work.

For the purpose of washing the gold from the worthless material, a pumping plant is necessary. Pumps for such purposes are generally of the centrifugal type, since they are capable of raising large quantities of water and, whenever necessary, loose dirt as well.

Appliances for entrapping and holding coarse gold and amalgamating fine gold, such as sluices and riffles, are necessary. The separation may be accomplished by blankets, undressed hides, sluices, riffles, or amalgamating plates, sometimes by all combined, depending on the fineness of the gold.

When work is prosecuted night and day, arrangements must be made for lighting. While some other good

system of lighting may answer, an electric arc-light plant should be figured in the specifications, as such lighting is generally the most satisfactory.

THE SUCTION DREDGE

42. The Centrifugal Pump. — **Suction dredging** depends on pumps. None are so suitable for this purpose as centrifugal pumps, which depend for their action on the pressure produced by the centrifugal force of a quantity of water rotated rapidly by the vanes of the pump; Fig. 17 is a view of a centrifugal pump with one half of the casing removed, so as to show the vanes *a* and their location in the pump. The suction pipe connects with that part of the case that has been removed and delivers the water to the vanes at their inner ends, near the hub *S*, the vanes being made sharp at this place, so as to offer less resistance to the entering water. The vanes revolve in the direction of the arrow, being driven by a shaft through the hub *S*. When the vanes are revolved, the air between them is driven out by centrifugal force, thus forming a partial vacuum. Water is forced in through the suction pipe by the pressure of the atmosphere and fills the space between the vanes. The water, of course, is made to revolve with the vanes, and the centrifugal action drives it outwards into the spiral-shaped passage *D*, which leads to the discharge pipe.

FIG. 17

The form of the vanes and of the passage *D*, Fig. 17, is of great importance in determining the efficiency of a

centrifugal pump, which are most efficient when working under low heads, and are seldom used for lifts greater than 40 feet. For low heads and large quantities of water, they give excellent results, and are especially useful when the water contains grit or other impurities that would destroy the pistons and packing or prevent the closing of the valves in other pumps. Since there are no valves or other restricted passages, centrifugal pumps have been largely used in dredging machines for pumping water containing large quantities of mud, sand, and gravel; and, in fact, anything can be pumped that will be carried through the pump and pipes by a current of water.

In Fig. 18 is shown the shell *a*, and open runner *b*, and a closed runner *c*. The closed runner discharges the entire run of the wheel and is considered by the makers to be more efficient than the open runner. It is further claimed that

Fig. 18

the closed runner wears but little, the wear coming on interchangeable balance rings, while the open runner wears rapidly and requires frequent renewal. Special pumps are sometimes made with only one passageway through the runner, which is large enough to allow the passage of lumps of material as large as the diameter of the pipes fitted to the pumps. The pump shown in Fig. 18 is driven by a belt,

FIG. 19

Fig. 20

but others are directly connected to steam engines or electric motors. A pump delivering 2,000 gallons of water per minute requires about 30 horsepower and is not, therefore, economical in the use of power, although very efficient in handling large quantities of water in a short time.

43. The Scow.—Fig. 19 shows the framework for the hull of a dredge. The ribs are firmly bolted to the bottom and deck timbers and these to longitudinal sills. The hull is constructed with 3-inch pine planks, planed to form close joints, which are afterwards caulked with oakum and tarred to make them water-tight. The scow is decked over with 3-inch planks in order to make a firm structure that will withstand the racking caused by the machinery in motion.

Fig. 20 shows a side view of the hull decked over with some of the machinery in place. The scow should set evenly in the water, and the machinery must be trimmed to that end; it is therefore usual to place the machinery that goes in the center of the boat first and then arrange the remainder to balance the craft. If this is not readily accomplished, stone ballast must be used.

44. Working a Suction Dredge.—The tail-pipe to the centrifugal pump is flexible so that it may be moved along the river bottom. The pump that is mounted upon the deck of the barge lifts material from the bottom to a sump either constructed in the barge or alongside; the latter situation is probably the best, as it can always be kept supplied with water from the river, by making proper connections. From the sump another pump raises the material to the sluices, or if necessary to a washing trommel; the latter, however, is seldom needed, since the pumps are fairly good washers; besides, in case the dredging pump should choke, the sluice pump can keep on working, water rushing into the sump from the river as fast as it is pumped out. It is a good plan to cause the material raised from the river to pass over bars or through a rotary screen to separate the coarse from the fine material and to permit only the latter

Fig. 21

to enter the sump; in such cases the coarse material is dumped over the side of the scow. The material flowing into the sluices passes over a series of riffles, in which the gold is accumulated, while the refuse passes over the stern of the boat. While this system is cheaper than some others, as far as first cost is concerned, it is not applicable to deposits whose bed rock is wavy or contains crevices, for the reason that no matter how strong the suction may be it will not be sufficient to lift heavy particles of gold that cling to the bed rock or lie in crevices.

45. Advantages and Disadvantages of the System. The machinery and appliances for suction dredging take up but little deck room and are not as extensive or heavy as those in other dredges, hence a lighter barge may be used. This system, however, has numerous disadvantages. The power required for pumping is necessarily large and the quantity of gravel and water brought up is not easily regulated. If gravel is too free, the tail-pipe is liable to become choked; and if tight, the suction does not lift it. Large rocks cannot be moved by it, and therefore accumulate in a layer, through which the effect of the suction cannot reach bed rock. After the gravel and sand have been brought from over the bottom, the pump cannot perform satisfactory work. Pump linings wear rapidly on account of the grinding action of the gravel and require frequent relining, which added to the cost of power make this system unsatisfactory. While suction pumps will dredge mud from harbors admirably, they are not as suitable for gold dredging as some other methods.

THE SHOVEL DREDGE

46. Steam-Shovel Dredging.—Fig. 21 shows a steam-shovel dredge at work. In this figure two barges are shown side by side, one for the machinery and the other for the sluice. The object of this is to avoid extra width and permit the sluice to be placed on either side of the dredge, and so work either bank of the stream. The machinery for

working this dredge is of the regular steam-shovel-bucket type. The scoop lip of the dipper *a* is reenforced to prevent its bending when coming against rocks. The doors *b*, through which the dipper discharges, are lined with leather or rubber to make as tight a joint as possible. The hopper *c* is made with screen-bar floor, and the material is washed down the sluice *e* by water coming from a pump through the pipe *d*. The coarse material that will not wash through the screen bars and into the sluice *e* is raked or pushed overboard. Riffles are constructed in the sluice. The dredge shown has been working a number of years satisfactorily on the Chestatee River, Georgia.

The shovel, or dipper, is operated by an arm *f*, that works in gearing on a boom *g*. The dipper moves up or down and swings back and forth, while the boom moves horizontally either right or left, both together describing an arc of a circle in order to come to rest over the hopper. In some instances the arms are made 20 feet long and the boom 60 feet. The hulls are made from 40 to 100 feet long, according to the design and power of the dredge. The capacity of the dredges varies also, according to size and power; some buckets hold 1 cubic yard and others 3½ cubic yards, hence the yardage excavated will vary from 500 to 2,000 cubic yards daily. The manufacturers' estimates are somewhat more than the yardage given. The dredges are fitted with steam winches for moving the boat from place to place.

47. The Clam-Shell Dredge.—Fig. 22 shows a river dredge fitted with a **grab,** or **clam-shell bucket** for excavating. The buckets are made in various forms with teeth, or serrated edges, to penetrate the ground. In whatever way they are constructed they must open and close automatically. The buckets are worked by machinery, but are hung from the end of a swinging crane. When at work they are dropped open into the water and their weight, with the momentum obtained in falling, drives their teeth into the river bed. The rope that raises them first acts on

the wings of the bucket in such a way as to draw the dirt between them into the receptacle that they form when

FIG. 22

closed. The bucket with its contents is then raised to the end of the crane, which now swings right or left to the desired dumping place. The buckets have a capacity of 2 cubic yards.

The hull for this dredge does not differ, except in particulars, from similar dredges. It has a swinging crane with suitable machinery for moving it about. The crane in the illustration is so arranged that it can swing in a complete circle, but this is unusual, 270° being the maximum swing, with a radius of 25 feet. The bucket delivers gravel to hoppers that may be on the barge with the crane or on a separate barge, as in the illustration. If on the barge with the crane, two hoppers will be needed, one on each side, situated sometimes as high as 14 feet above the deck. The speed of excavating and delivering material to the hoppers is about 40 buckets per hour; from this and the capacity of the buckets the quantity of material treated daily can be estimated.

This system has some commendable features. The material is delivered moderately dry to the hoppers and there is

(a)

(b)

FIG. 28

no danger of the banks caving, which sometimes interferes with the workings of suction and bucket elevator dredges. It may also be used for dredging in deep places. There are, however, many disadvantages. It frequently happens that a stone lodges between the jaws of the bucket, thus preventing their closing completely, in which case the bucket reaches the hopper only partly full. The daily capacity is greatly reduced in this manner. The crane requires the constant attention of one man, and as the work is exacting, three shifts a day are necessary. The cost of working is about five times that of a good ladder dredge, being over 50 cents per cubic yard.

BUCKET DREDGES

48. Introduction.—The **bucket dredger,** in general, is shown in Fig. 23, (*a*) being a plan and (*b*) an elevation. Several firms manufacture bucket dredges that differ chiefly in the construction of the links, buckets, and washing machinery. Fig. 23 (*b*) shows the hull *a* of a barge, which is squared at one end, but catamaran-shaped at the other. The buckets *b*, which with their links form an endless chain belt, move around two tumblers situated at the ends of the ladder *c*. The ladder is raised or lowered by wire ropes fastened to it and passing over pulleys in the gantry *d*, thus connecting with the drum of a steam crab. This arrangement permits the ladder to be lowered between the catamaran for deep dredging or be raised for shallow dredging. The buckets attack the bank *e* when in motion and raise the material to the hopper *f*, where they discharge. From the hopper the material is washed into the trommel *g*, and is further washed by streams of water coming from the pump *h*, and entering the trommel through the pipe *i*. The fine material passes from the trommel on to the tables *j*, and goes from there to the sluice boxes *k*, where the gold is retained and the refuse washed overboard. The coarse material passes out of the trommel into a trough *l* leading

to a boot; from which it is lifted by the elevators working on the boom *m*. The latter may be raised so that the material may stack to a height of 30 feet if desired. In the plan, *a* is the deck of the barge, *b* are the buckets, *d* the gantry, *f* the hopper, *g* the trommel, *h* the pump driven by belt, *i* the pump pipes leading to the hopper and to the trommel, *j* the gold-saving tables, and *k* the sluice boxes. The chute *l* leads to the stacker *m*, which projects over the stern of the barge; *n* shows the boiler, with steam pipe leading to the

Fig. [illegible]

engine *o*, that furnishes power for moving the machinery, and also for moving the boat from place to place. At the corners of the barge are hawse holes *p*, through which anchor ropes pass to the winches. These are worked by levers in such a way that each winch is independent of the others. One winch raises the bucket ladder, another the staking boom, while the other four move the boat from place to place. A better view of the gold-saving tables, settling boxes, and sluice box, is shown in Fig. 24.

49. Buckets.—In Fig. 25 is shown a series of buckets *a*, which are joined by means of links *b*. These buckets and links are made very strong in order that they may be able to lift a heavy load and sustain considerable strain without parting. It will be noticed that every other link carries a bucket and that the fastenings to the links are made as nearly grit-proof as possible, besides being so arranged that they may be quickly removed and another immediately substituted in case of breakage. Each bucket is reenforced by a heavy piece of steel *c* for the purpose of saving the bowl of the bucket from wearing; hence whenever this heavy lip wears out it may be replaced and the main part of the bucket thus preserved.

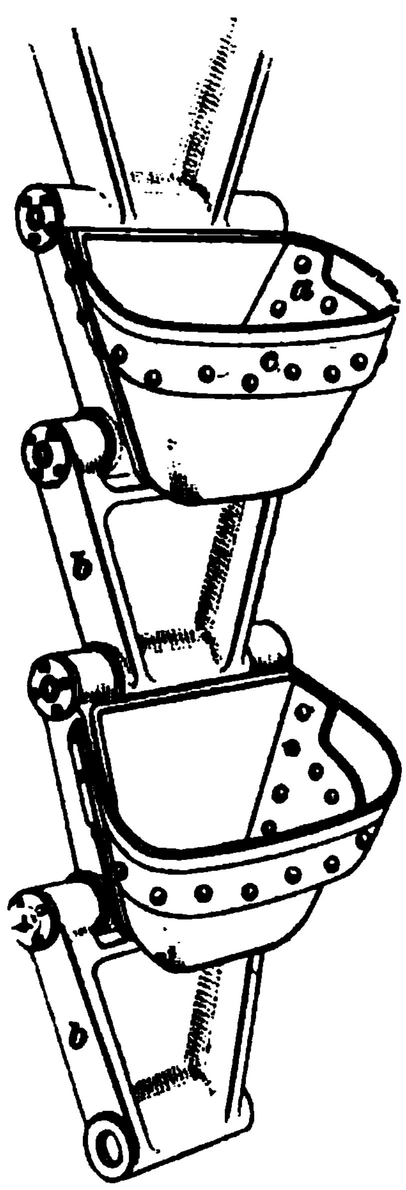
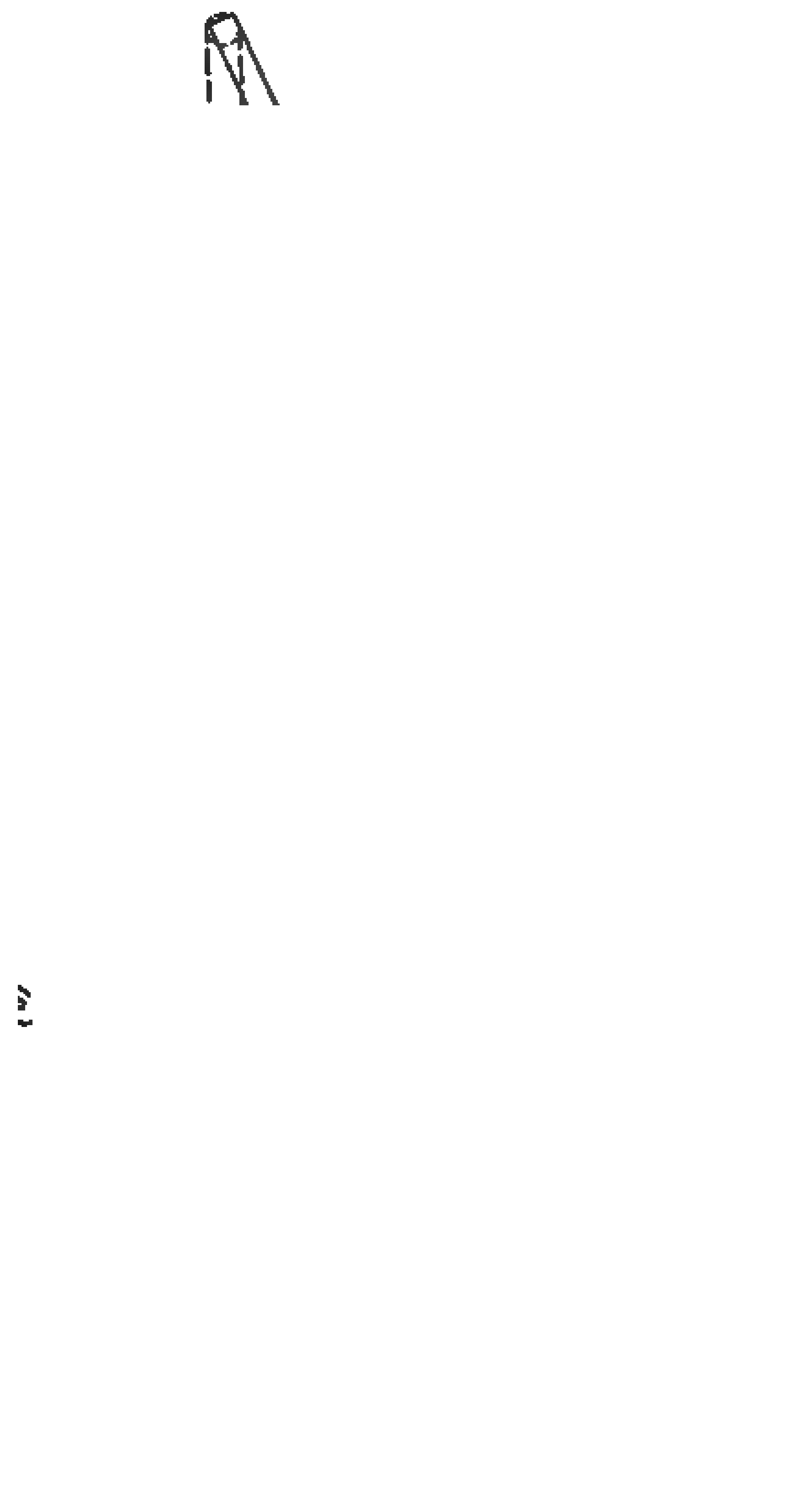


FIG. 25

The dimensions of the chain and buckets depend on the size and the capacity of the dredge for handling material. Buckets vary in capacity from 2 to 18 cubic feet, the latter being approximately of sufficient size to carry 1 ton of material. The capacity of the dredge will not only depend on the size of these buckets but also on the speed with which they travel, and the capacity of the washing and sluicing apparatus. Buckets having 2 cubic feet capacity are supposed to excavate and raise 1,000 cubic yards per day of 24 hours. Buckets of 18 cubic feet capacity are supposed to excavate and raise 8,000 cubic yards of material in 24 hours.

50. Electrically Driven Bucket Dredge.—Fig. 26 (*a*) shows a type of dredge that differs somewhat from the one described. The material is excavated and conveyed by the bucket belt *a* to a shaking screen *b*, upon which it is dumped. The fine ore is allowed to pass the screen while the coarse material is carried by the conveyer belt *c* to a stone chute *d*,

and is discharged from this into a hopper *e*, from which it is delivered on to the tailings stacker *f*. The fine ore after



(a)

FIG. 26

passing through the screen is led over the gold-saving tables *g*, and into a spitzkasten *h*, and thence to the tailings

stacker. The tables are given an inclination toward the spitzkasten. In the plan, Fig. 26 (*b*), *m* is the motor that drives the excavator, not shown in the plan, the screen *b*, the stacker *f*, and the winches *w*; *c* is the belt conveyer for removing the material that passes over the screens to the stone chute *d*; *e* is the hopper into which the large stones are dumped; *f* is the tailings stacker; *g* the gold-saving tables; and *h* the spitzkasten.

For gravel containing both clay and boulders a double-lift dredge may be used, the material thus receiving an extra handling, which effectually breaks up the clay and separates the gold.

51. Ordering Dredges.—For designing dredges, the following data should be known. The character of the ground to be excavated—whether it is sand, sand or gravel, sand, gravel, and boulders, or gravel and clay-cemented material. The buckets, elevating machinery, and screens will be constructed upon this data. The depth of the deposit below water level and the thickness of the deposit to bed rock should be known, for on this information the length of the barge and excavating arm or ladder will depend. If the gravel is loose and easy to dig, a medium weight of machinery may be used; but if very hard, a heavy and strong outfit will be required. If clay is present, a double-lift dredge may be necessary. If there are many boulders, the buckets must be exceptionally large and the grizzly very strong. Should the gold be medium or coarse the sluice box will save it; but if fine, the table system is better.

DRY-PLACER MINING

52. Introduction.—There are localities where ancient rivers have eroded gold-bearing rocks and deposited the gold, with other debris, where now scarcely any water is to be obtained, at least not nearly enough for hydraulicking. There are three characters of placers of this description,

which are known as **dry placers.** The first have no water and hence are not worked. The second has water near by,

Fig. 27

which, however, cannot be used for hydraulicking on account of a lack of fall; these may be worked by inland dredges,

FIG. 28

Fig 29

that is, scows fitted up in the same manner as river dredges. The third has some water but not sufficient to use for dredging with scows, and must be impounded and used over and over again.

53. Floating Inland Dredges.—Fig. 27 shows a dry placer with the floating dredge to the left; this dredge was built on the ground. In the foreground is shown a boring machine testing the value of the ground. Fig. 28 shows the completed dredge in operation, having dug its own pond for floating in. The water is seen entering through a ditch *a* at the right of the pond, and the barren excavated material *b* is shown stacked up in the rear. When the ground within the radius of the arm is worked out, the boat is moved to another position. After the gravel has been removed to bed rock, it may prove worth while to shut off the water entering the pond, pump out the pond, and clean the bed rock by hand. Machinery is never as effective in cleaning bed rock as men with shovels, but if hand work is to be done, the tailings, which take up about 1½ times as much room as the ground in place, are to be discharged outside the pond.

54. Dipper Dredges.—The dipper is particularly useful in inland dredging, since it will handle large and small material in a very satisfactory manner. Fig. 29 shows a machine digging its own pond; the bucket is under water but the bucket arm *a* is shown. The scow is held steady by the spuds *b* at the corners. The stream being discharged from the pipe *c*, at the rear of the scow, consists of water and fine material, that has passed over the sluices and into a sump. The coarse material is removed by the stacker belt *d*, which in this case is a rubber belt provided with flights. Wherever possible, such belts should be used, as they are light and serviceable. The hopper in this illustration is not shown as it is on the opposite side of the dredge. The platform is constructed for the foundation of another hopper, in order to have one each side of the barge.

55. Traction Dredges.—This form of dredge is useful where very little water is obtainable or where suitable dump room for the waste material cannot be had. It is

FIG. 30

also used for removing overburden from auriferous gravel. These machines are self-propelling and are capable of handling a boom of great length, enabling them to dump

FIG. 81

the material into a sluice on the bank if it is desired to do so. The dredging machinery is placed on a strong car running on a track and while working is held in position by means of jack-screws.

Fig. 30 shows a traction dredge in connection with which a small car is used for carrying the material to the sluice. The dipper *a* that is operated by the boom *b* excavates the material and dumps it into a small skip car *c*. The skip carries the material to a hopper in the top of the dredge, where it is automatically dumped and passed over grate bars, the coarse material going to the dump, while that which passes through the grizzly falls into a revolving screen *d*, in which it is washed and sized, the coarser going to the stacker *e*, and so to the dump in the rear, while the fine material goes through the screen and is washed through the sluice box *s*.

This dredge is capable of handling from 800 to 1,500 cubic yards of material in 10 hours, the bucket having a capacity of 2 cubic yards. For a bucket having from 3 to 3½ cubic yards capacity, from 1,500 to 4,000 cubic yards may be handled in 10 hours of continuous work. Water that is necessary for these dry-placer dredges may be impounded and, with a slight addition of fresh water, used over. The dirt is washed both in the hoppers and in the trommels, so that all clay balls are thoroughly broken up.

56. Clam-Shell Bucket Dredgers.—In Fig. 31 is shown a clam-shell bucket dredge at work in dry-placer mining. The closed bucket at *a* is lifting overburden from a placer deposit; the bucket *b* has discharged its load and is ready to fall and grab another. On account of the length of the boom and the heavy load coming at its end, a dry-placer or traction dredge is placed on four trucks, one near each corner of the car platform. This arrangement requires double tracks but gives stability; however, to further add to the stability and prevent the platform rocking and injuring the machinery, jacks are placed at the end corners where the boom swings. When

washing machinery is connected with traction dredgers, it may be placed on a separate car alongside the excavator, or it may be arranged on the same car by extending hoppers over the sides. In the latter case both ends of the car must be jacked.

PNEUMATIC JIGS

57. Dry Washers.—Where there is no water, placer mining is a poor proposition; however, it is carried on in several localities by means of machines termed **dry washers.** These machines depend on air to separate the barren from the valuable material, but at the outset they are confronted with the problem of sizing. It is useless to attempt the concentration of ore in a dry way by machine until it has been sized, and as soon as one considers sizing machinery, the factor power becomes of the uppermost importance.

Power can be obtained from gas engines and in localities where there is a little water such engines may make it feasible to work the ground. There are several dry-placer machines that are worked by hand, and are said to save 63 per cent. of the gold.

DEEP PLACER MINING

58. Drifting.— In the case of deep placers overlaid with lava caps, **drift mining** is followed in order to obtain the material on the bed rock. In some cases, only the bed-rock gravel will pay for excavating; while in others it is possible to work the material for some distance above bed rock, and occasionally it is found possible to work the gravel on two different levels, provided a false bed rock existed some distance above the true bed rock. In working drift mines, the entrance may be effected either through a shaft or tunnel. In some cases an old river bed is cut by a modern river bed in such a manner that a drift can follow the old bed, but in other cases it will be found necessary

to drive a cross-cut tunnel from an adjacent valley or to operate the mine through a shaft. Operating through a shaft has the disadvantage that the water must be removed, and this often renders the mining operations unremunerative.

Fig. 32 gives a plan and two sections on the Sunny South drift mine in California. This mine was operated through a tunnel driven from an adjacent valley in such a way that it passed under the lowest part of the ancient river bed, and access was obtained to the gravel by means of raises. Drifting is only profitable when the precious metal has been

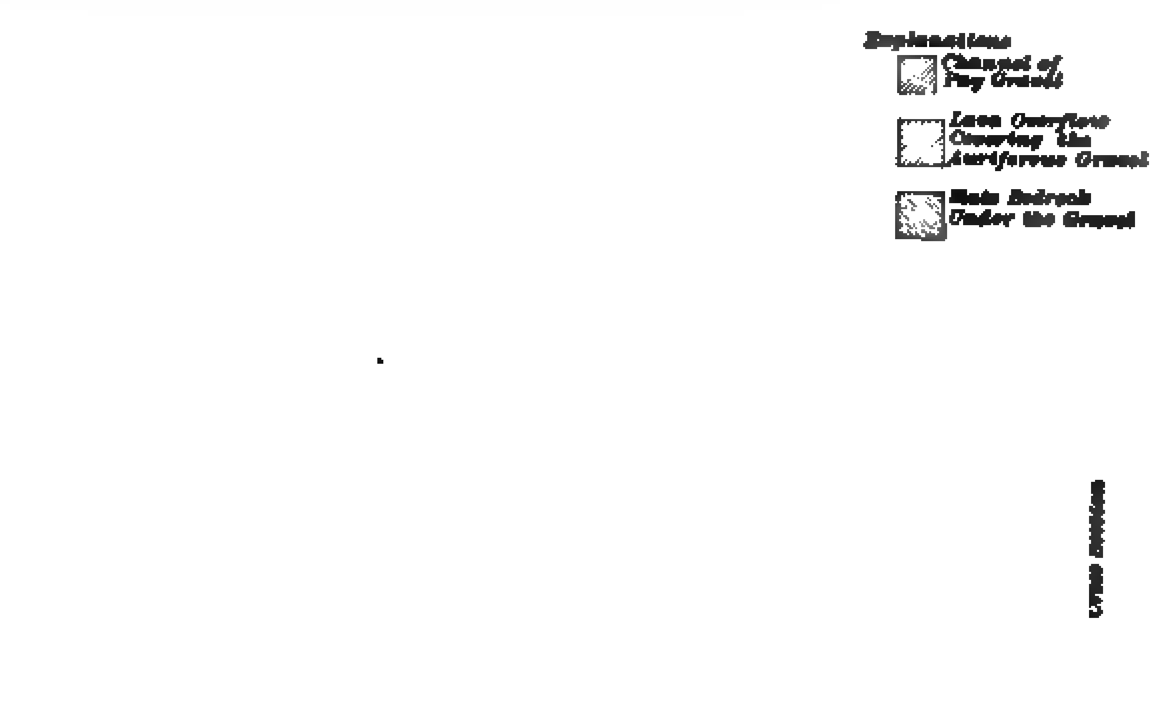


FIG. 32

concentrated in well-defined strata or portions of the channel. The direction of the tunnel through the "rim rock" is a matter of great importance, and should be located so as to reach the lowest part of the deposit, otherwise it will be necessary to pump water and raise gravel to the tunnel, and the main objects sought in drifting will be missed.

59. Handling and Treating the Gravel.—The gravel is removed in mine cars to the mouth of the tunnel, where

it is dumped and washed in sluices, or if the material is firmly cemented it is crushed by machinery, previous to washing. When stamps are used for crushing they differ from those in gold and silver mills in that they have a double discharge and a very much coarser screen, it usually being at least $\frac{3}{16}$-inch mesh. The greater part of the gold is amalgamated in the battery, but copper plates outside of the battery are also employed, and these are usually followed by sluices. As a rule, no attempt is made to save the gold-bearing sulphides contained in the gravel. In some mines, steam locomotives are used for transporting the men and materials through tunnels that are over a mile in length, but arrangements must be made in such cases for ventilation, otherwise the gases from the locomotive will smother the men.

60. Timbering and Mining.—The tunnel is secured, like an ordinary mine drift, by means of timber sets similar

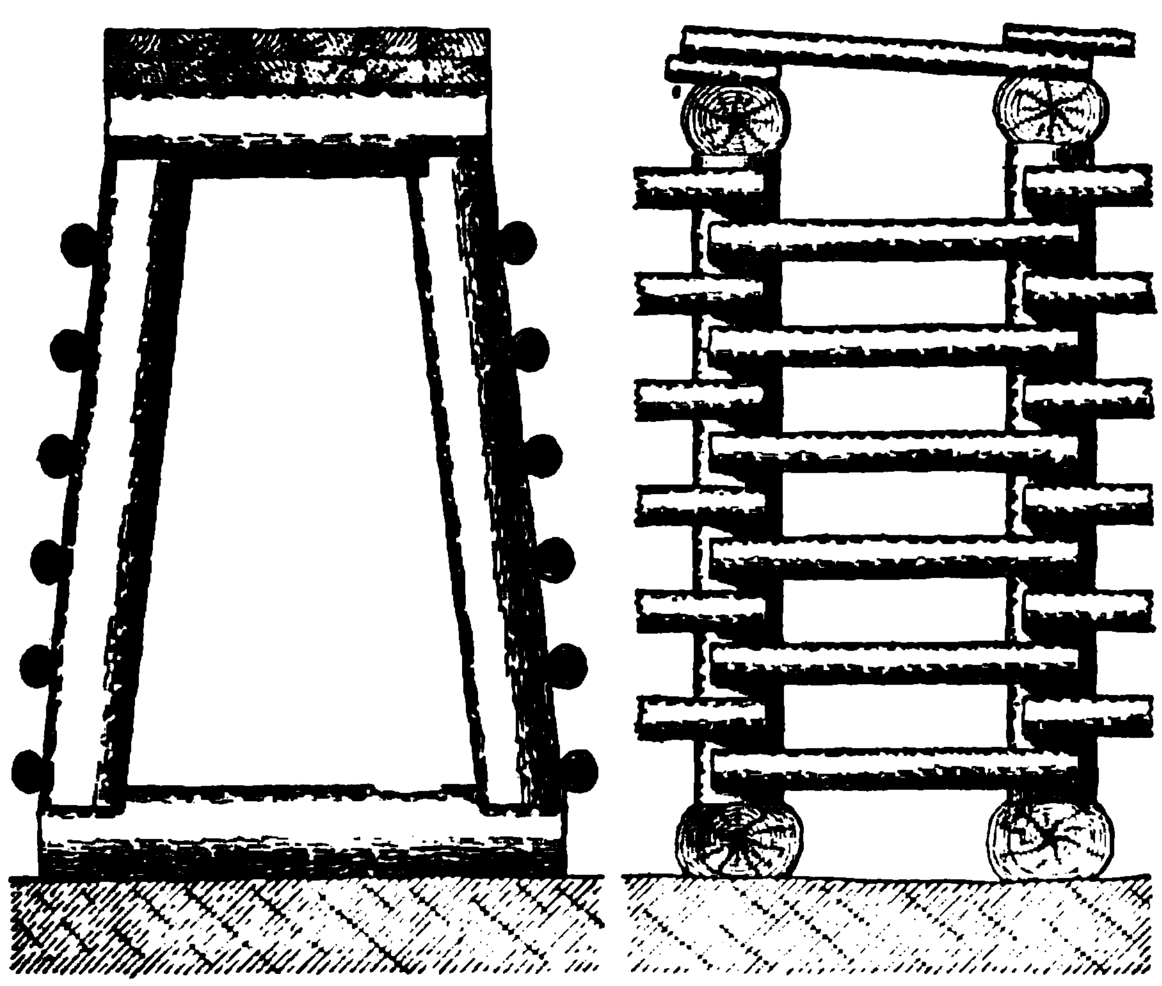


FIG. 33

to those shown in Fig. 33. The deposit is divided into panels or divisions, which are subsequently worked either

on the pillar-and-stall system or by means of square work. When worked by pillar and stall, the pillars are sometimes robbed clean and the entire deposit on bed rock recovered. This is accomplished by building packs of the larger boulders in the worked-out rooms to replace the pillars and also by supporting the roof temporarily by means of posts and caps similar to those shown in Fig. 34. If the gravel is comparatively soft, it may be necessary to timber the entire roof, in which case the posts may support stringers, which in turn carry lagging; or a series of drift sets may be used side by side.

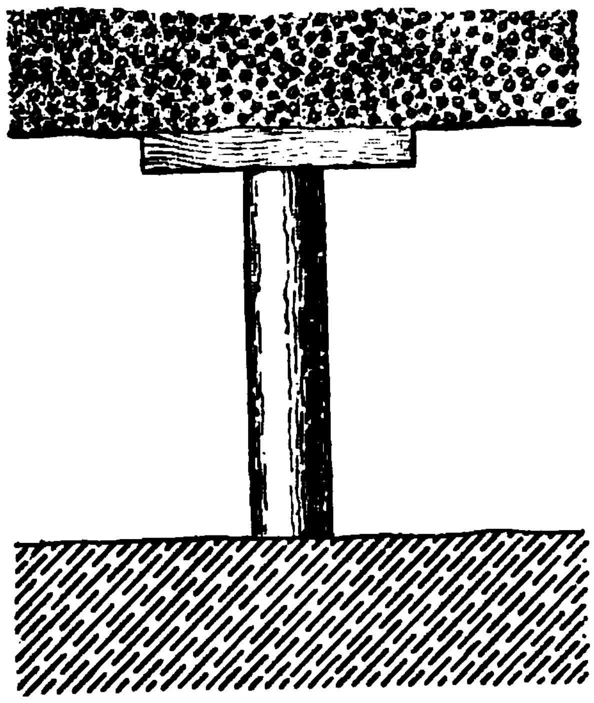


FIG. 34

When the square-work system of mining is followed, the drifts and breasts are carried in both directions at right angles, leaving the pillars as illustrated in Fig. 35. It will be seen that by this means three-fourths of the deposit can be recovered, providing the drifts and pillars are of the same width.

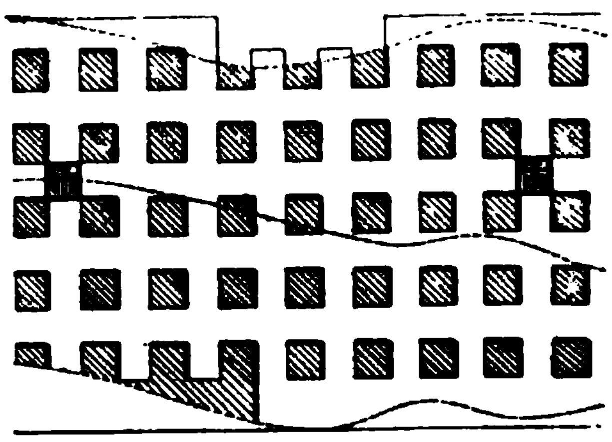


FIG. 35

61. Tunnels and Outlets for Hydraulic Mines.—Some placer deposits occur in ancient river beds, where they are not overlaid with lava caps, but are so situated that they have no natural dumping ground. An outlet for these deposits is frequently provided by driving a tunnel to some neighboring ravine or cañon. A shaft is sunk to connect with the inner end of the tunnel, and the gravel washed down through the shaft and out through the tunnel. Sometimes sluice boxes are placed in the tunnel, while in other cases the material is allowed to run on the rock-bottom floor of the tunnel, which then forms a natural ground sluice. When this latter plan is followed the material is liable to wash the

floor of the tunnel into irregular hollows, and may in time cause very serious caves. There is one advantage in having a sluice inside of a tunnel, and that is the tunnel can be provided with doors, which can be kept locked in such a manner as to prevent men stealing the gold that has accumulated in the sluices. After the work has progressed for some time and the gravel has been removed to bed rock, at the point where the shaft descends to the tunnel, it may be possible to place a portion or all of the sluices on the bed rock above the point where the material enters the tunnel.

62. **Blasting Cement Gravel.**—When working **cement-gravel mines** by hydraulicking, it is frequently necessary to loosen the gravel by means of blasting. For this purpose a tunnel is driven into the bank some distance, and drifts turned to the right and left, parallel to the face. Boxes or barrels of powder are packed into these

FIG. 36

drifts and connected with one another and with a surface battery by means of wires, so that they can be exploded when desired. The tunnel is then carefully sealed with fine gravel or clay, so as to prevent the escape of the gas from the explosion through the tunnel opening and thus increase the efficiency of the explosive. The tamping should be thoroughly rammed in with wooden mauls. Occasionally blasts are fired by means of fuses; when this is done, two or three lines of fuse should be laid, so as to avoid misfires.

The general arrangement of the tunnel drifts and charges is shown in Fig. 36, in which *a d* are the wires connected with the two poles of the battery, and *b* are the charges to be fired. Comparatively hard cement gravels are blasted by means of black powder, which is simply used to lift and somewhat disintegrate the bank. Where the cement gravel

is extremely hard, it is blasted by means of low-grade dynamite or giant powder. In banks of ordinary cement gravel, 50 to 150 feet high, the main drift should be run in a distance of two-thirds the height of the bank to be blasted. The cross-drifts from the end of the main drifts should be run parallel to the face of the bank, their length being determined by the extent of the ground to be removed. From 10 to 20 pounds of black powder is required per 1,000 cubic feet of ground to be loosened. Even when black powder is employed for the blast, the exploders *b* are usually inserted in cartridges of giant powder and placed on top of the paper covering the black powder. By this means a much more powerful detonation is obtained, and the action of the black powder is made more effective.

63. Working Frozen Ground.—In Siberia and Alaska the ground is frozen to a considerable depth, and the summer season is so short that as a rule, it does not thaw sufficient gravel to give the miner a fair season's work; in some cases it never thaws to bed rock. On this account the material is mined during the winter while frozen, and is thawed and washed during the summer. The frozen gravel is much harder to work than ordinary rock, owing to the fact that it is so tough that it resists drilling, blasting, or picking. The miner therefore thaws the ground before attempting to break it down. He accomplishes this by sinking shafts and building a fire against a portion of the ground to be removed. In sinking the shaft, if the surface is frozen a wood fire is made, which thaws the ground for a little distance. The fire is often rendered more effective by a cover of charcoal, to confine the heat. When the fire dies down, the miner scrapes aside the embers, and shovels away the thawed ground beneath until he comes to a frozen portion, where another fire is built and the operation is repeated. This is continued down to bed rock. The sides of the shaft are given what support is necessary by means of timber cribbing or rough square sets with lagging. From the bottom of the shaft a drift is started, every foot having to be thawed. A strong

wood fire is built against the face of the drift and covered with charcoal, as before, and allowed to burn out. After the fire has burned out, the thawed material is removed and another fire built. All workings must be tightly, though not necessarily heavily timbered, owing to the fact that the constant use of the fires underground soon softens the roof, and portions of it are liable to cave.

Some simple ventilating device is usually necessary to remove the gases generated by the fire. This may be accomplished by means of a brattice until the mine has been extended a sufficient distance to drive an air shaft. The work is usually carried on by means of the square-work system, as shown in Fig. 35. The effect of the fires in the drifts is to raise the temperature to an oppressive point, so that in some large Siberian mines the miners work naked, though the temperature outside may be several degrees below zero. An amount of wood equivalent to the thickness of 1 foot across the face will thaw about the same depth of gravel, and 14 inches is practically the maximum depth that can be thawed with one fire.

Another method of thawing the ground is by heating boulders and dropping them into the shaft, where they are left until the ground about them is thawed, when they are removed and reheated and the thawed dirt shoveled out.

64. Thawing by Steam. — The system of thawing ground by fires has been almost entirely superseded by the use of steam except for prospecting purposes. Steam is used as follows: A boiler house is built close to the mouth of the shaft in which a boiler is placed with a 1-inch or 2-inch steam pipe running down to the bottom of the shaft, where a T manifold is attached. To this T manifold are fastened several lines of steam hose, and to the end of each hose a steel nozzle called a *point* is fastened. This is a strong pointed steel tube having three small holes bored about 1 inch back of the point. A heavy nut is screwed on the other end so that the tube can be driven into the ground

with a mallet; the hose is attached to the nut at right angles to the tube. The tube is placed in position by driving the point into the dirt about 6 inches. Steam is then turned on and the point driven in as it thaws its way. If large rocks are met with, it is sometimes necessary to drill the hole. The steam remains turned on and the points kept in place for 8 or 10 hours, when they are removed and placed in another section of the mine. If the dirt thawed is allowed to stand for a day, it will impart its heat to a much larger area and thus save considerable time and work.

Fig. 37 (*a*) illustrates the method of thawing by steam. *a* is the boiler where the steam is generated; it should not

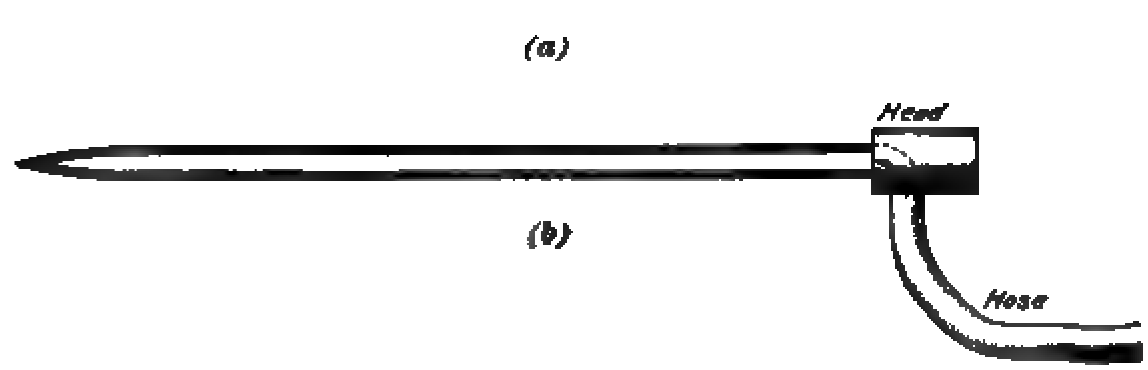


FIG. 37

be placed directly over the excavation, as the ground is liable to cave in summer; *b* is a steam pipe, from 1 to

2 inches .n diameter, through which the steam is conveyed to the manifold, shown at *c*, from which it is distributed to the hose *d* leading to the points *e*, which are driven into the gravel bank. Fig. 37 (*b*) shows the point, which is from 3 to 5 feet long and 1½ inches in diameter.

SURFACE ARRANGEMENTS AT ORE MINES

GROUND PLANS

ECONOMY IN HANDLING PRODUCTS

1. Mine Openings.—The majority of mines are opened, with the least amount of trouble, by small tunnels or shafts, and it is not until a mineral deposit is known to be valuable that permanent mine openings are located and plans formulated for working on a large scale. During prospecting work the crudest devices are adopted, which gradually give way, as the property develops, to more economical methods of handling material. The prospector's opening is made where the deposit can be most readily attacked; but the permanent opening should be located with a view of economy in mining, in transporting ore and supplies, and in treating the ore.

Comparatively few ore mines become large producers, that is, have an output of 150 or more tons per day; but most mines ultimately require larger power plants than were used in their early days, if the same output is to be continued from greater depths.

At most mines, there will be accumulations of low-grade ore and tailings, which cannot, for the time being, be made commercially valuable, owing to the lack of necessary facilities. But these accumulations or dumps, having cost money to mine, should be cared for until they are extensive enough to warrant an expenditure for their treatment.

Numerous cases can be cited by old mining men where neglect to care for such lean ore and tailings has involved great loss to the mine owners; and other cases can be cited where dumps have yielded large amounts of money.

2. Plan of Surface Arrangements.—Ore mines are seldom so located that all conditions are favorable to the economical handling of ore or the arrangement of buildings; therefore, those dispositions which have a direct bearing on mining and hoisting ore should take precedence over dispositions connected with shipment or ore dressing and milling.

When planning surface arrangements for hoisting ore, it should be remembered that an increase of power will probably be necessary sooner or later. This increase should be anticipated; that is, the initial installation of hoisting machinery should admit of enlargement as fast as an increasing output of ore calls for it. To mine low-grade ore profitably, a larger plant must be installed than is needed for high-grade ore. There are two reasons for this: larger quantities of low-grade ore must be mined in order to meet fixed charges; and low-grade ore occurs in larger masses than does high-grade ore, and the valuable parts of it are more uniformly and regularly distributed.

The arrangement and location of the hoists should be followed by arrangements for furnishing the mine with supplies and for assorting the ore, particularly if the mine is on a side hill, or is in some high, narrow valley where teams must be used for moving supplies and ore to and from the nearest railroad station. Mills, shops, and other structures must be situated as close to the mine as possible, although frequently it has been found more economical to transport the ore some distance than to place the mill near the mine opening.

Owing to the fact that surface conditions at each mine are not the same, it is impossible to give more than a general idea of a mine plan; consequently, greater attention will be given to the machinery and the transportation facilities in use at actual mining plants than to an elaborate discussion of ideal mine conditions.

HOISTING PLANTS

3. Power and Hoisting.—The structures needed for a hoisting plant are a house for boiler and engine, and a head-frame. The introduction of machinery in mining presupposes the certainty of abundance of work, and a profit over and above the cost of installation and operation. The capacity of the hoisting machinery is governed by the depth of the mine, the number of stopes, and the quantity of ore each stope can produce; yet it is always good policy to allow for extra power to meet demands consequent on development and mining at greater depths. To replace the old hoisting engines with new ones in order to obtain increased power is often unprofitable, for the fixed charges connected with the cost of mining increase with the depth, and thus the mine is less able to bear additional expense than it was at the outset.

Provided the contract price of mining remains stationary, the only way to reduce the fixed charges is to increase the output; from which it follows that, if the ore becomes leaner, an increased quantity must be mined in order to meet the fixed charges and still show the same profit. Either case warrants the installation of more power than is required at first, since the extra output can be readily applied to increase the profits.

Another advantage in connection with having extra power available at mines lies in the fact that, even if the hoister does not require all the power for ore raising, the surplus may be made available for any sudden increase of water that the pumps cannot handle.

4. Arrangement of Hoisting Machinery.—There are two systems used in arranging hoisting machinery at mines; namely, the *individual* and the *central system*. In one case, there is one engine and one drum to each shaft; in the other case, one engine operates several drums. The economy, if there is any, in a central plant, consists in concentrating the power in one large engine, instead of distributing it among several small engines.

5. Individual Hoisting Plants.—Where the **individual system** is used, the shaft may have a large and continuous output, in which case the engineer must be at his engine during the entire hoisting period, while the fireman attends to the boiler and oils the machinery. At a small mine where the output does not exceed, say, 50 tons daily, the engineer can do the work of both fireman and engineer, as in such cases the hoisting is more or less intermittent, and no great loss is sustained if a stoppage for 5 minutes occurs once in a while. Under such conditions the boiler and engine should be placed as near together as possible, so that the engineer can see his steam gauge without making a special journey to the boiler to look at it.

At the majority of ore mines, even when the shafts are deep, or where there are several shafts, hoisting is done with one cage rather than in balance. In most cases, the output is not large enough to make much difference; but in other cases there is good reason for using two cages. The hoister at small individual hoisting plants is usually enclosed in a house built against the head-frame, with the boiler room off to one side. When the engine is large, the mine deep, and the head-frame high, the engine house is separated from the head-frame, and also from the boiler house. At such mines the boiler power is required for pumps and air compressors as well as for hoisting, and to avoid fire risks the boiler house should be separated from the engine house and head-frame.

6. Economy of Individual Hoisting Plants.—Where one cage is hoisted at a time, there is not much opportunity to practice economy, at least from a mechanical standpoint, except in the first cost of rope, machinery, and apparatus. By hoisting from a compartment shaft having two cages, there is an economy in fuel that may offset the cost of additional wear on the apparatus. Where there is a large output, the work is concentrated and loss of time is minimized by two cages and an individual hoist. The largest mines are equipped with individual plants, although there may be several

hoisting engines and shafts. At anthracite mines and at the Lake Superior copper mines, the speed of hoisting is so great that there is considerable strain on the engineer's nerves, for which reason he is relieved every few hours.

7. **Central Plants.**—The other system followed in arranging the hoisting plant—the **central system**—consists in placing a number of drums in one building and connecting the ropes with the various shafts. This system

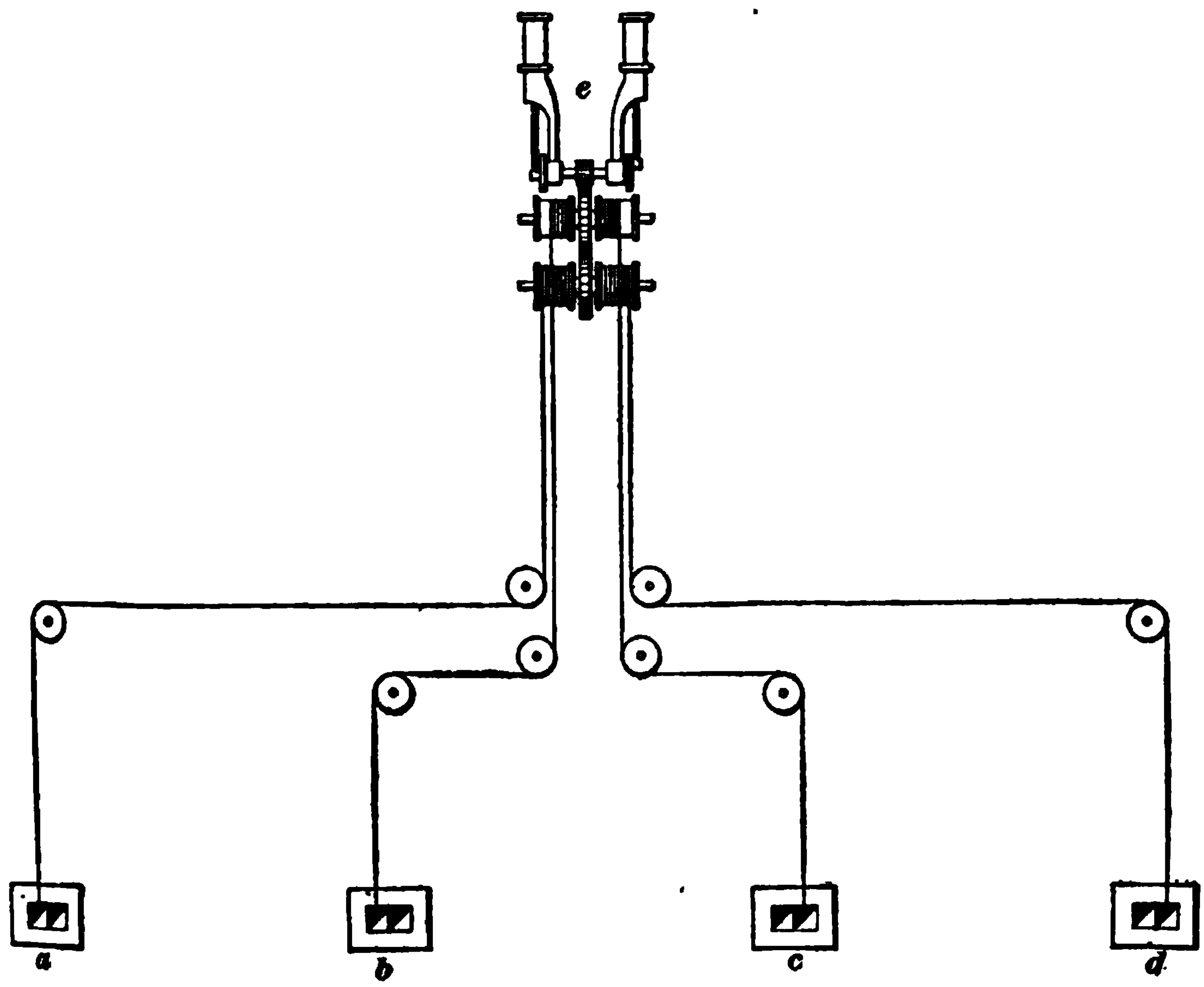


FIG. 1

is especially applicable to mines where the output from the different shafts is more or less irregular, and where most of the shafts have but a single cage or skip, which works out of balance. Fig. 1 illustrates a plant laid out on this line. There are four shafts, *a*, *b*, *c*, and *d*, which are operated by drums placed at *e*. One of the advantages claimed for this system is that in case of fire at one of the shaft houses the machinery will not be destroyed, and work may be carried on from the other shaft houses. This is really no advantage, since the engine house, or possibly one of the shaft houses,

may catch fire. The advantages consist in economizing steam by having the boilers near the engine, and in the economy that always accompanies concentration of labor.

8. **Head-Frames.**—The kind of head-frame used for a permanent plant will depend on whether the mine shaft is vertical or inclined. In the former case the head-frame is placed directly over the shaft; in the latter case it is placed some distance from it, so as to conform with the inclination of the shaft near the surface. These head-frames are constructed with reference to the strains caused by the engine pull on the hoisting rope and the weight of the load and rope in the shaft. Just before the load begins to move, these forces are equal, and the direction of their resultant is a line bisecting the angle between the two parts of the rope and passing through the center of the pulley and the center of gravity of the head-frame.

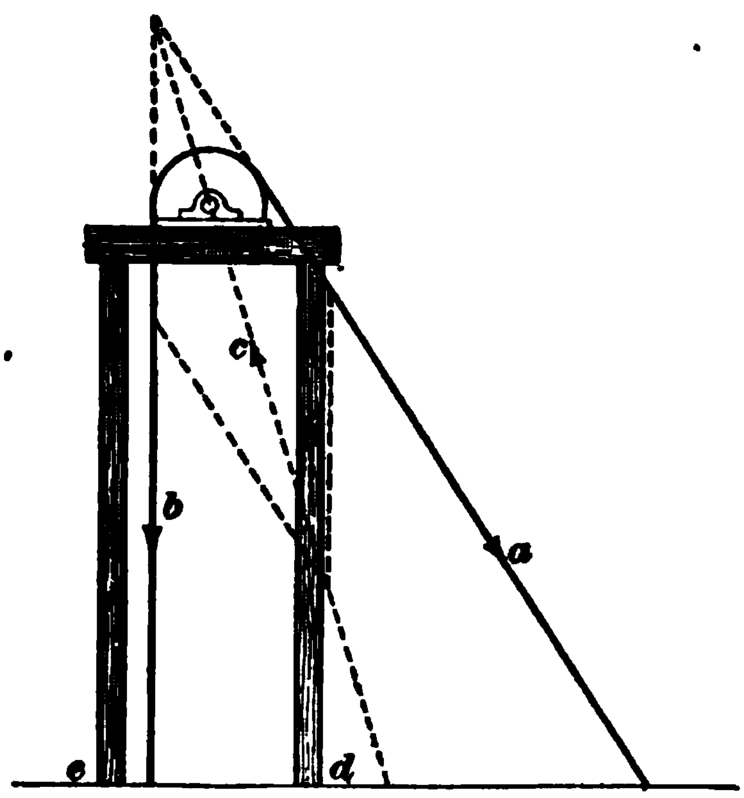


FIG. 2

In Fig. 2, the engine is pulling in the direction *a*, and the combined load, cage, and rope in the direction *b*. The resultant *c* of the two forces when extended falls without the center of gravity of the frame in such a way that the tendency is to rotate the frame at the point *d* and raise the legs of the frame at *e*. This form of head-frame is therefore to be avoided as unstable.

Fig. 3 shows a better arrangement for a head-frame than Fig. 2, since there is a brace to give stability to the legs. If *a* represents the line of pull from the engine, and *b* the line of pull due to the cage, load, and rope, then *c* represents the resultant of these two forces, which falls within the triangle formed by the leg *e* and the brace *d*. This being within the center of gravity of the frame, the latter has no tendency

to turn on the point *d*. Many head-frames are constructed in this manner. It is apparent in Fig. 3 that only the legs *e* and the braces *d* are necessary for this head-frame, since the front legs are not subjected to either strains or pressure. In the gallows frame and the Montana frame the pulley wheel is so placed that the resultant of the two forces *a* and *b* passes through the brace *d*. The two front legs are then discarded,

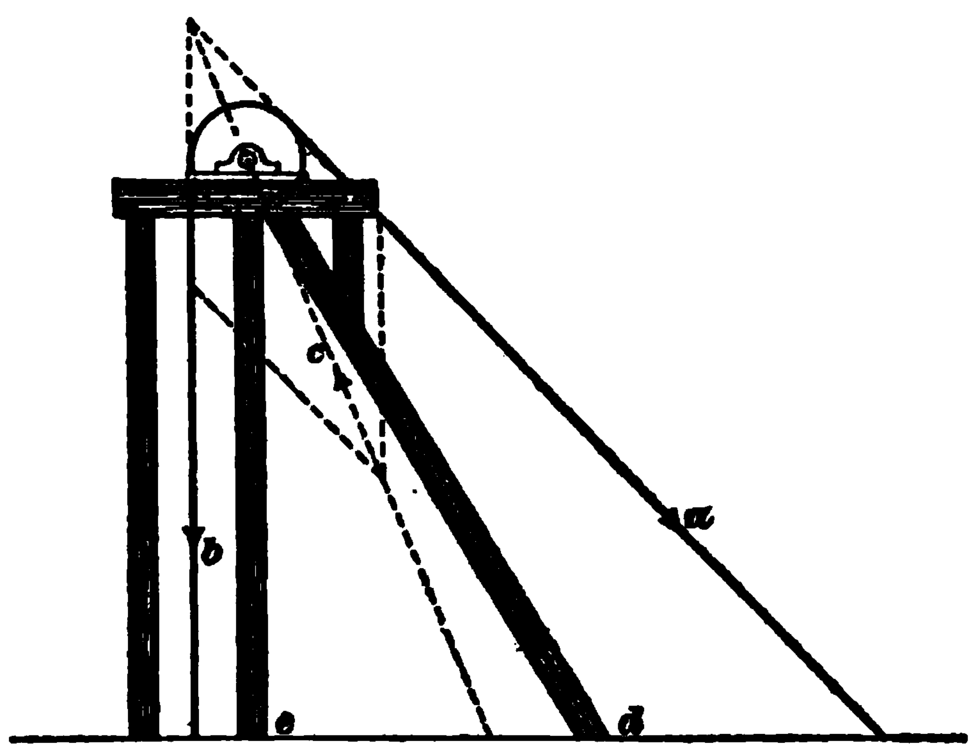


FIG. 3

the two legs *e* placed so as to form an inverted **V**, with the apex near the pulley wheel, and the two braces *d* spread at the ground and brought as near as possible to the wheel at the top of the frame. If the journal-boxes can be placed at the top ends of the braces, so much the better; if they cannot, then they should be placed in line with the resultant, as shown in Fig. 3.

9. **Slope Head-Frames.**—The head-frame in Fig. 4 is supposed to have a hoister in line with the rope *a*, and to sustain a skip being drawn up a slope by a rope represented by the line *b*. The resultant of the two pulls is *c*, which falls within the center of gravity of the frame. In this case, *b* is not a brace against the hoisting pull, but is a brace against the lowering pull. The figure is not a complete illustration of a slope frame; usually, the cap piece on which the pulley wheel rests is continued to the left over additional posts ranged on that side. There is no danger of

such a head-frame being pulled over, since the strain of the load is always in the direction of the arrow *b*.

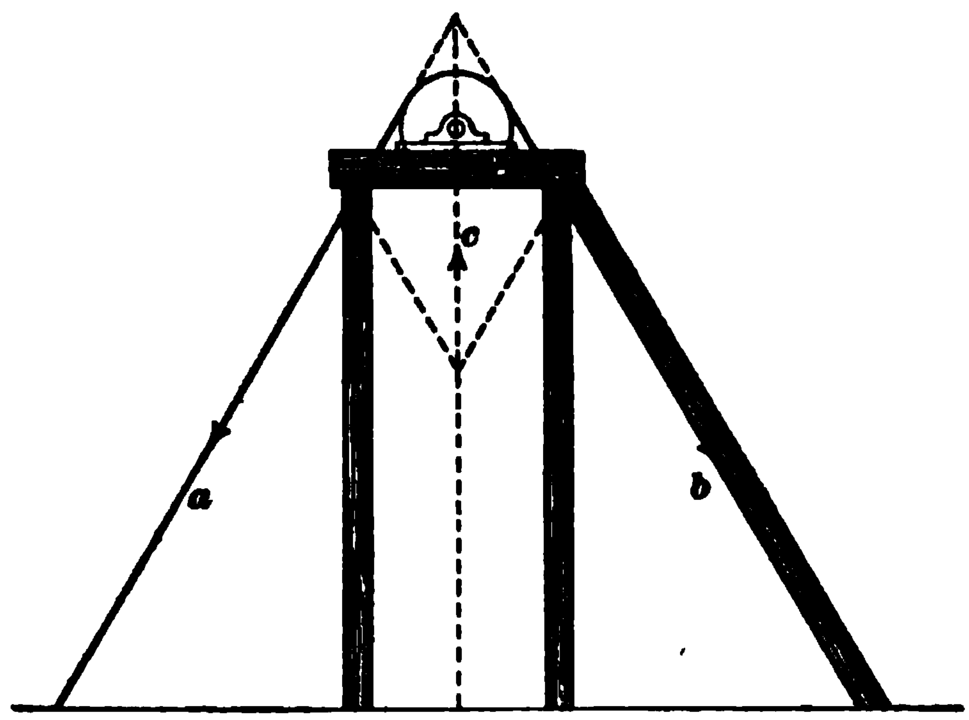


FIG. 4

10. Head-Frame Construction.—The principle involved in the design of head-frames is to transmit the compressive strains so that they will fall within the center of gravity of the structure. If the working strains on the sheaves can be transmitted in a straight line to the foundation of the back braces, they will be within the center of gravity of that structure, and both rigidity and economy in material of construction will be obtained. It has been assumed in the structure of head-frames that the pyramid is the most stable form, and the sheaves have been so placed as to avoid all the eccentricity possible, firmness being secured by back braces and stiffeners. At first, it was common to see rectangular four-post frames with braces; then, four-post frames with the posts spread so as to act as legs and braces at the same time; and, finally, the gallows frame, consisting of two posts and two back braces to receive the compressive strains.

These principles are well known among engineers, but surface conditions frequently prevent the construction of the ideal head-frame; hence, four posts and braces are often to be seen where two posts and braces would seem to be all that is necessary. The frame properly belongs to the footwall side of the vein, although in some instances, owing to surface conditions, it is placed on the hanging-wall side.

11. Wooden Head-Frames.—The posts and braces for wooden head-frames should be of sound timber, free from shakes and knots. If it is not possible to obtain timber of a size suitable for posts and braces, two timbers that united give a sufficient area of cross-section may be bolted together as shown in Fig. 5.

The compressive strength of timber varies, but almost any first-class timber affords sufficient compressive strength along the grain to withstand the strains placed on it when hoisting. The lengths of the sticks composing a head-frame are usually such that they must be braced to prevent their bending. Thus, in Fig. 3, there should be a strut for the brace *d*, to act as a stiffener. The heel of the strut should come near the foot of the post *e*.

FIG. 5

In case there is difficulty in obtaining timbers sufficiently long for posts and braces, two timbers *a* and *b*, Figs. 6 and 7,

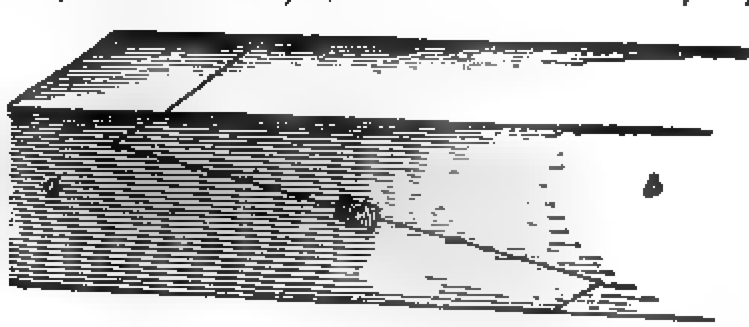


FIG. 6

may be scarfed and held together by keys *c*. Such joints must be braced and strengthened by fish-plates when used as

FIG. 7

members of a head-frame. The bolted joint shown in Fig. 8 is as strong as any for posts, provided the stress passes

uniformly along the center of the posts *a* and *b*, and their ends are butted together flush, and held by the fish-plates *c*.

Fig. 8

The various timbers composing a head-frame are generally mortised and tenoned, and then fastened together with tree-

Fig. 9

nails in such a manner as to form a solid structure comparatively free from vibrations when under working stresses.

12. **Heavy Framing by Cutting Joints.**—Since head-frames require careful construction, and since the engine house or the ore bins are frequently built on to them, the subject of framing timbers must be considered here.

One form of heavy framing is illustrated in Fig. 9. The heavy sill timbers *a* are secured to the timbers *b* by notching, as shown in the figure, and by the use of drift bolts, in order to preserve the whole area of the timber in undiminished strength. The posts *c* are fastened to the timbers *b* by means of tenons and wooden treenails or iron pins. The timbers *e* and *d* for the second floor are united by notching and by pins, while the posts *f* for the next story are secured by tenons and treenails. The braces *g* are sometimes notched into the posts and sills and secured by heel tenons and pins; at other times, they are tied with a straining piece *h*, Fig. 11, spiked to the timber frame. (Fig. 10 illustrates a tenon on the end of one of the braces.)

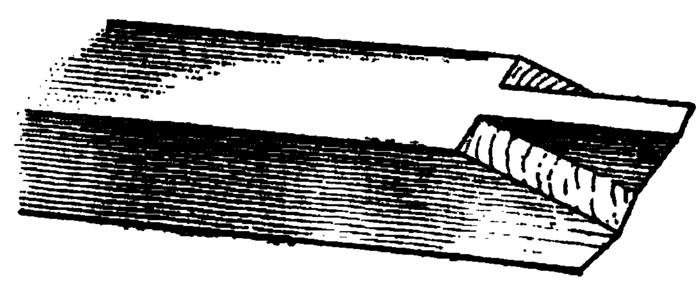


FIG. 10

The floorbeams *i*, Fig. 9, may be notched to the same depth as the sills *b*, so that the floor *k* can be laid on top of the floorbeams and the sills *b*. Where the braces do not extend the full width of the post, they can be placed either flush with the outside of the building, as shown, or centrally on the timbers. The manner of placing the braces flush with the outside, as illustrated, has the advantage that, whatever form of siding is employed, it will be secured to both the posts and the braces, and will thus aid in stiffening the building.

13. **Heavy Framing Without Cutting Joints.**—Fig. 11 illustrates another system of framing, which is to a large extent on the balloon-frame order, for it has no mortise-and-tenon joints and very little framing of any kind, the object being to obtain the full strength of the timber. The sill timbers *b* may be notched to *a* and secured by means of drift bolts; the posts *c* may be slightly notched into the timbers *b* and secured by drift bolts. The braces *g* are simply

pieces of plank cut and spiked as shown in the illustration, the pieces *h* being spiked against the posts or sills and filling the space between the ends of the braces. Where the floor-beams *m* rest on the sill *a*, it may be necessary to use a narrower piece between the braces, as at *i*, or to saw notches into the piece *i*, against which the beams and braces can be

FIG. 11

placed. The timbers *d* and *e* for the upper floor of the mill are fastened together and to the posts *f* and *c* by slight notching and drift bolting. In some cases, no notching even is done, the posts simply being sawed off square and secured with large spikes or drift bolts. The floor *k* in this style of construction is laid exactly as in the previous case.

14. Corbels.—Where it is necessary to join sills or large horizontal timbers on the tops of posts, corbels may be used, as shown in Fig. 12. The corbel *a* is bolted to the two timbers *b* and *c*, and the post *d* is usually tenoned into the corbel, while the post *e* may have a wide tenon and be secured to the timbers *b* and *c* by means of two treenails or pins.

FIG 12

15. Combination Head-Frames.—One of the complicating factors entering into the subject of head-frames is the emptying of the cars as they come from the mine. Skips, automatic dumping cages, or buckets discharge the ore in the head-frame, and this necessitates ore bins for its storage, and possibly other arrangements also for assorting.

An important consideration in the design of a head-frame is its height, for, in most instances, rails, timbers, and pipes must go through the shaft into the mines. Rails are usually made in 30-foot lengths; only specially made rails come in smaller lengths. Steam or air pipes 2 inches in diameter average about 18 feet in length; hence, they do not need to be taken into account so much as do the mine rails; but the height of the head-frame above the landing platform must be at least enough to permit the handling of pipes of this length.

It is good policy, whenever surface conditions will permit, to place a combination boiler and engine house at least 60 feet away from the head-frame, so that in case of fire both buildings need not be jeopardized. At large plants where there are several boilers in use, the boiler house should be independent of the head-frame, which may then include the engine house. Where the head-frame, ore bins, picking

tables, and engine are under one roof, and that roof is over or in close proximity to the shaft, there should be two outlets to the mine, so as to afford a means of escape for the miners in case of fire in the head-frame. The laws of some states compel mining companies to have two openings to their mines.

16. Independent Head-Frames.—Where the loaded mine cars are hoisted from the mine, it is possible to so arrange things that the ore may be trammed directly to the ore bins or mill. In such cases, head-frames may be independent of ore bins, the mills or shipping stations instead being supplied with them. To the extreme right and left of Fig. 13 shaft head-houses *a* are shown to be connected by a trestle *c* with the mill *b*. These independent head-frames have their own hoisting engines, and when the cars are brought out of the mines they are trammed over the trestle *c* to the

FIG. 13

mill and their loads dumped. This illustration is taken from a photograph of a zinc plant in the Joplin, Missouri, district. The waste-rock dump is shown at *d* and an ore pile at *e*; the tower at *f* contains a bucket elevator for raising tailings from the mill and carrying them out over the trestle *g* to the dump. The tailings are automatically dumped, as shown by the white pile *i* at the back of the shaft head-house on the left. There is a settling pond *m* in the foreground, the water being used over again after passing through the mill.

CONSTRUCTION OF INDIVIDUAL PLANTS

LOADING AND UNLOADING FIXTURES

17. Fig. 14 shows a section of an individual plant, with a gallows head-frame *a*, shaft *b*, pumping engine *c*, hoisting engine *d*, and boiler *e*, all under one roof. This plan is not good, since the house covers the shaft to no purpose. If the building had been cut off at the line *x y*, the lumber and construction bill would have been reduced one-half and the danger to the miners from fire minimized; besides, better ventilation would be had in the mines. Such little covering as would be needed at the head-frame for the top landings is insignificant in comparison with this housing. Fig. 14 is intended to show the various important parts of an ore plant; there is no good reason why the pump-bob and head-frame should be housed.

18. Fig. 15 shows an individual plant that is at the same time a combination plant and assorting house. The illustration is taken from a photograph of a large gold mine in South Africa. It will be noticed that the buildings are situated well back from the mine mouth and that the skip *a* dumps automatically into an ore chute *b*. The waste rock is separated from the ore and both are trammed from the building, the former going to the dump and the latter to the stamp mill. The arrangements at this mine, owing to the contour

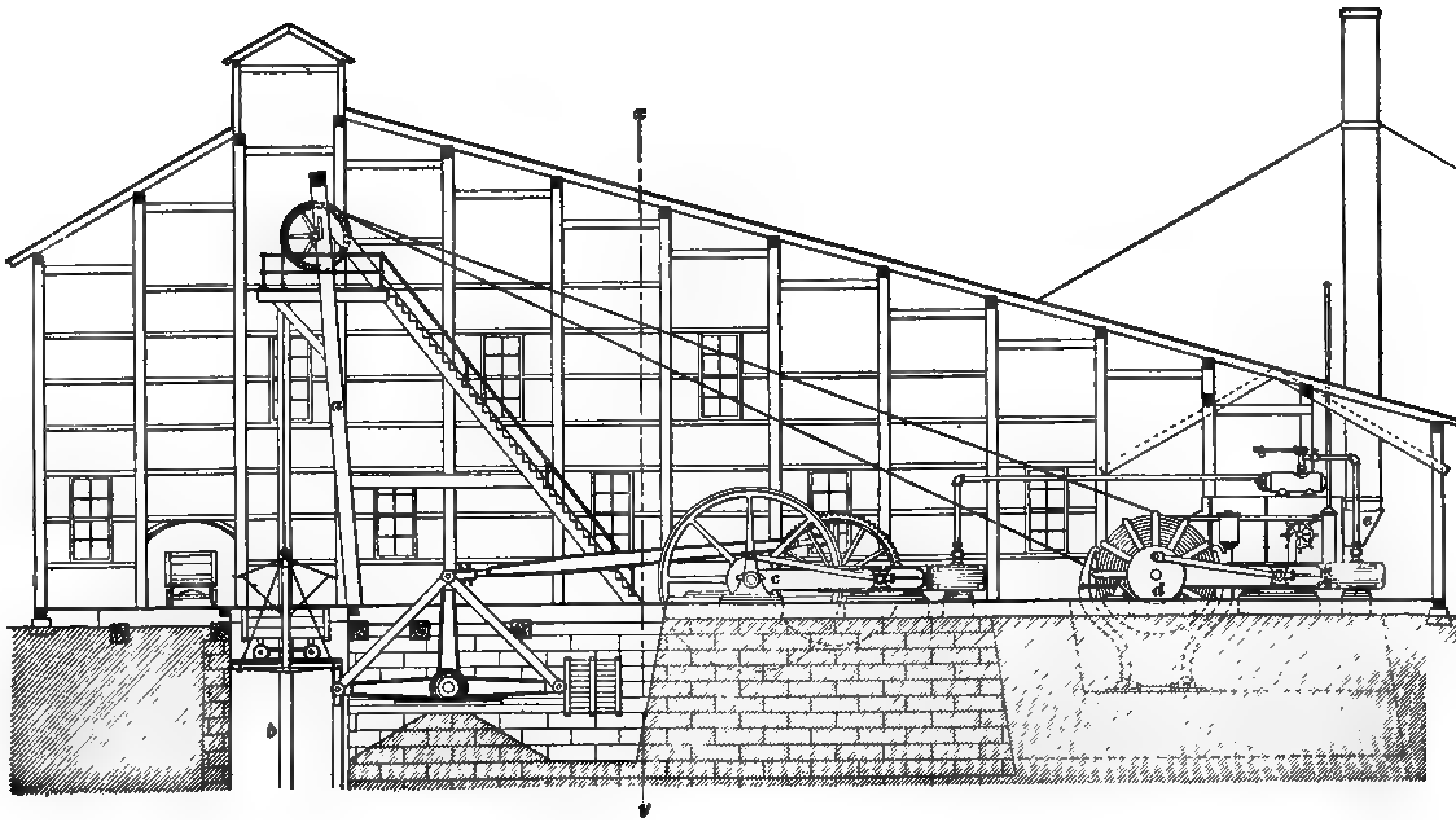


FIG. 14

Fig 15

of the ground, are particularly good. Every opportunity to take advantage of irregularity of surface should be made the most of. After the skip *a* has dumped its load automatically into the ore bin, the fine is separated from the coarse ore and rock, and the latter is conducted to the crushers, then to the picking tables, and so on to the bins, from which the material is loaded into the cars going to the mill.

FIG. 16

19. Head-Frame and Ore Bin.—Fig. 16 shows the head-house at an iron-ore mine. The slope mine has a skip road *a*, up which the skip runs to the automatic dumping arrangement *b*. At *b* the skip dumps its load on to a picking table, where the poor ore and rock is separated from the

good ore. The good ore is sent to the ore bin *c*, from which it is loaded into the railroad cars *d*. The waste rock is run in tram cars over the trestle *e* to the rock dump *f*, while the lean ore is moved in cars and stacked in a pile not shown. At such mines, in the Lake Superior iron-ore regions, ore is mined and stacked in winter, so that heavy shipments can be made by lake steamers during the summer months.

Fig. 17

20. Another illustration of a wooden shaft head-house and ore bin is given in Fig. 17. In this, as in Fig. 16, the skips are hoisted in balance on tracks *a*, the engine house being situated some distance away from the head-frame, as shown by the ropes leaving the sheave wheels *b*. The ore

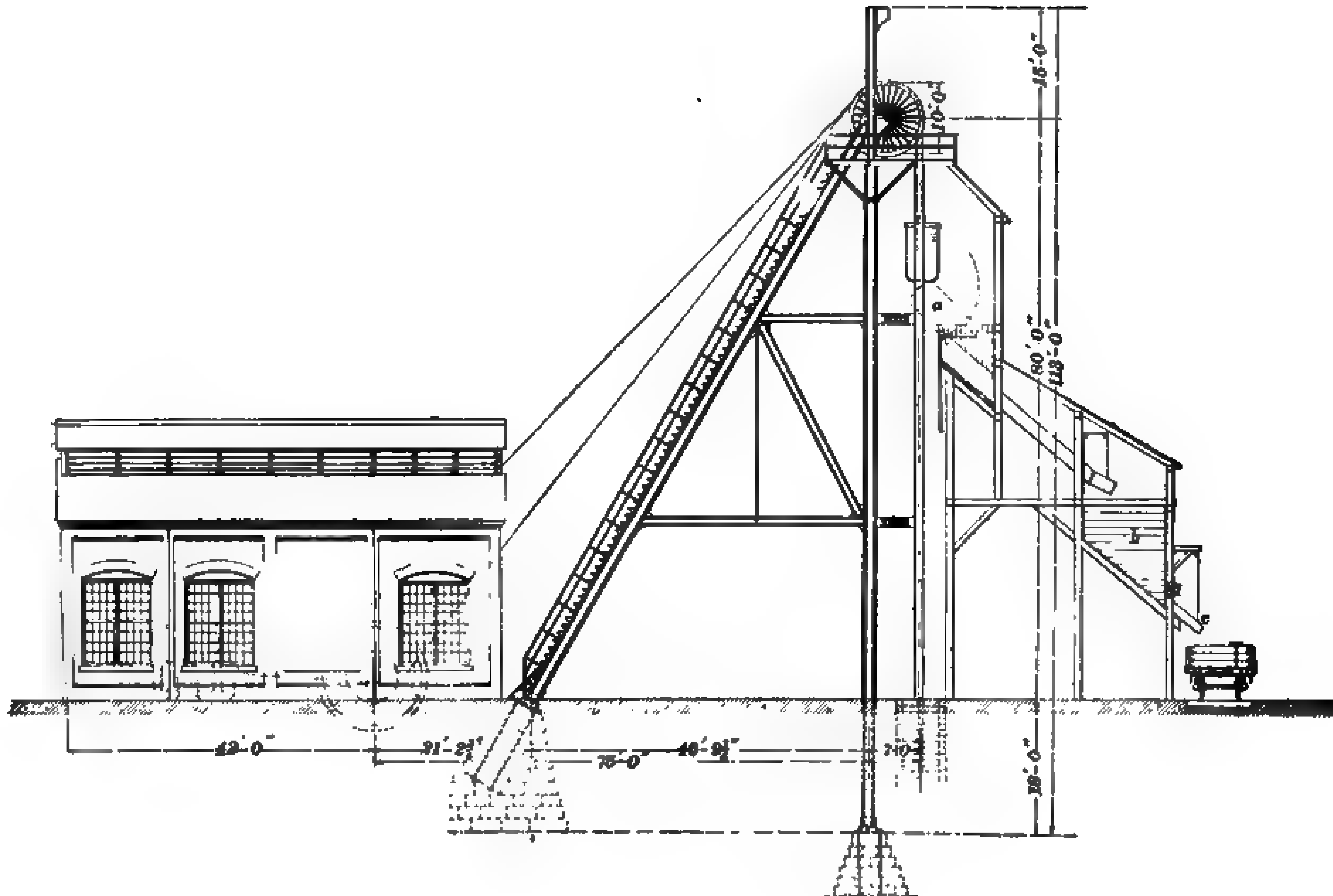


Fig. 16

bin is planked in at c, and has its floor sloped to the center loading gate e on one side, and toward a similar gate on the other side (not shown), from which are loaded the cars that run over the trestle d to the ore stock pile.

21. **Steel Head-Frames.**—Fig. 18 shows a steel head-frame with ore chute that was erected at a mine in Western

FIG. 19

America. The ground about the engine house and car track was made even by filling in; hence, the peculiar appearance of the masonry pillars, which are on bed rock. The ore, which is hoisted in specially constructed self-dumping buckets a, passes through a chute into pockets b, and then into the cars through the apron c. This arrangement makes

it possible to fight fire in the shaft, and, as the engine house is of brick, there is little danger to machinery, head-frame, or men. This style of head-frame is becoming quite common at large mines, on account of its safety, strength, and durability. If, however, the sheave-wheel boxes had been placed directly on the ends of the back braces, it would have been better. There being but two light posts, much less material is used than in most steel head-frames.

Fig. 19 shows a steel head-frame constructed at one of the deeper mines in South Africa. Two cages are used for hoisting mine cars, the ore being trammed from the shaft to the mill in the same cars that bring it from the mines. The sheave wheels are placed in the proper position, but the object to be attained by having four posts is not clear. The weight and stability of such head-frames permit rapid hoisting, while the fact that they are not enclosed at the surface allows a free circulation of air for ventilating purposes. The construction of this head-frame is similar to that of the Montana head-frames, as they are called in the western parts of the United States; they combine maximum strength with minimum structural material when the posts and legs are properly spread at the base and focused at the sheave wheels.

22. Sheave Wheels.—Sheave wheels are either loose or keyed. Where the hoisting rope is wound on a drum, the sheave wheel must have a small fleeting movement on its axle in order to keep in line with the coil on the drum as much as possible, and thus prevent undue wear and riding of the rope. The method of fastening such sheave wheels to head-frames is shown in Fig. 20. The two stringers *a*, which carry the wheel and axle, are provided with pillow-blocks *b*; the axle is provided with collars *c* to prevent it from working out of the boxes. While the hub of the wheel turns on the axle, the axle turns in the boxes, by this means lessening friction. The keyed wheel, however, centers

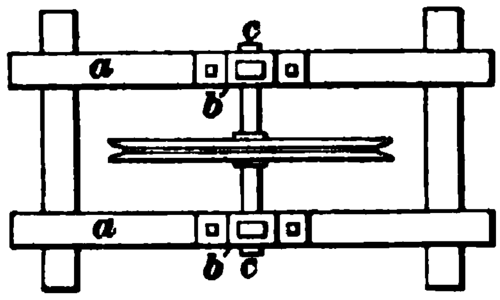


FIG. 20

over the shaft at all times, and turns with the axle. It is mounted on the head-frame in a similar manner, the only difference being that the stringers *a* are placed nearer the wheel. In case a keyed sheave wheel is used, the drum must be placed some distance away from the sheave to prevent the rope from riding and avoid excessive friction. Sometimes, fleet wheels are placed between the drum and the sheave, to prevent sagging and reduce the tendency of the rope to ride when winding on the drum. At many plants, flat ropes with reels centered on fixed sheaves, as at *d*, Fig. 14, are used in preference to the round rope and loose pulleys.

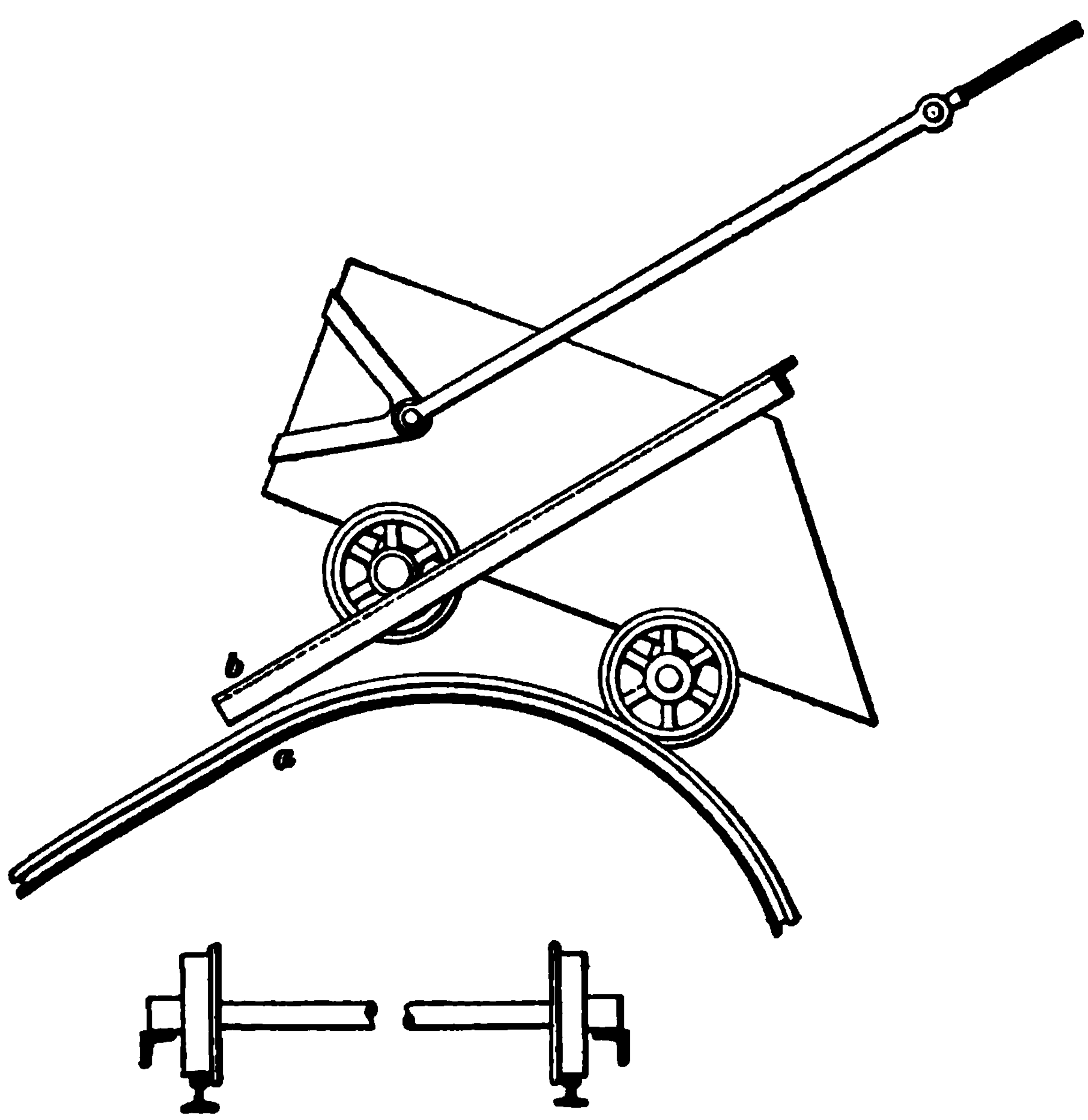


FIG. 21

23. Dumping Skips.—It is usual to adopt some system of automatic dumping where skips and buckets are used for hoisting. One of the best arrangements for slope skips is that shown in Fig. 21. The rear axle of the skip is extended a few inches as shown, so that it will run along the angle iron *b* while the front wheels of the skip follow the curved track *a*, which is a continuation of the slope track. This

raises the rear of the skip and lowers the front end, thus giving an angle sufficiently great to permit the ore to slide out.

24. Bucket Dumps.—Self-dumping buckets have not come into very general use, but several patented arrangements have proved satisfactory, and will, therefore, be described here. The automatic dumping device shown in Fig. 22 has a frame with a track *a*, on which travels a trolley *b* carrying the hoisting sheave. When the bucket *c* coming from the shaft *d* reaches the proper position, the carriage moves up the inclined track to the position shown

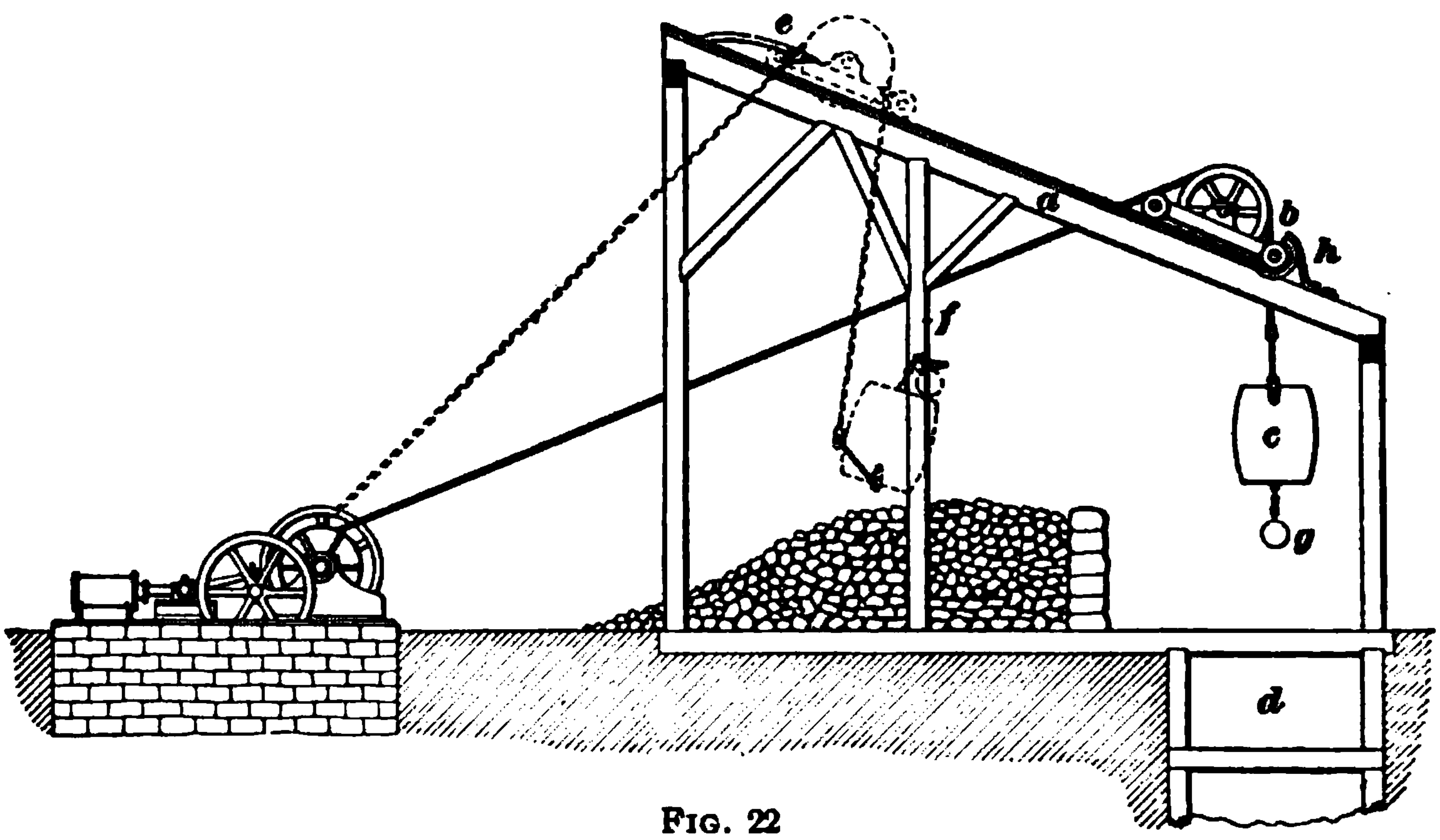


FIG. 22

by the dotted lines, where it is stopped by the hook *e*. Then an arrangement on the post *f* engages the chain *g* hanging from the bottom of the bucket *c*. This having been done, the rope is slacked and the bucket discharges by turning upside down. When the bucket is empty, the hoist pulls it up until it reaches the trolley, when the hook *e* is released and the bucket and trolley move down the incline until the curved-rail stop *h* arrests the trolley, and the bucket goes down the shaft.

25. Automatic Bucket Dumper.—Fig. 23 is a perspective view of a patented dump. The bucket *a*, after it leaves the shaft, moves up the hinged skids *b* until it reaches

the hook *c* at their top. The rope is then slacked, and the weight of the loaded bucket throws the skids out of balance and compels them to carry the bucket to the position shown by the dotted lines. As soon as the loaded bucket has been discharged, a slight pull on the rope, the bucket being empty, will return the skids to their original

FIG. 23

position; the bucket then returns to the mine by gravity.

26. Grizzlies.—If, when the ore is dumped, it is desired to size it for picking or crushing, it is passed automatically over a set of bars such as those shown at *A* in Fig. 24. That which passes through the bars into the ore bin *B* is ready for treatment; that which passes on to the floor *C* is to be cobbed and sent either to the rock dump or to the crusher.* The arrangement shown is for ore bins

from which ore is shipped to mills; but where large quantities of ore are handled, the arrangements are much more elaborate, and include picking tables or belts, for rock should never be shipped any great distance from a mine.

27. Ore-Picking Arrangements.—The proper place to dress ore for subsequent treatment is at the mine, since every pound of rock shipped adds to the handling and transportation charges. Fig. 25 shows arrangements for handling ore previous to shipment and for the disposal of waste rock.

Fig. 25

The head-gear *a* is shown at the top of the head-frame. The fine and coarse ore coming from the mine is separated by a grizzly, the fine going to a picking belt and the coarse to a rock crusher situated in the building *c*. Both fine and coarse ore eventually reach the picking belt—which is also a traveling belt—which should not move faster than 45 feet per minute. The belt is housed in the structure *d* that connects the ore bins *b* with the crusher house *c*. As the belt moves from *c* to *b* the ore it carries is hand-sorted, the rock picked out being thrown into rock chutes, which deliver it to a scraper conveyer *e* that carries it to the rock dump.

28. Picking Belts.—Where ore is assorted for shipment, it may not always be economical to install a picking belt, in which case picking tables are required; but where there are large quantities of ore to be assorted belts should be provided. The picking belt is shown in Fig. 26. It is not a new idea. It was adopted in 1876 at ore-concentrating and smelting works in Lake City, Colorado, where it was installed as a conveyer and automatic feeder of ore to

Fig. 26

crushing rolls; and in this capacity its further value as an assorting or picking belt was developed. It is only in recent years that its value as a means for assorting and conveying ore has been fully established. The ore as it is carried along is cobbed with a hand hammer on the belt. One of these belts is said to have conveyed 350,000 tons of heavy crystalline ore, in pieces about 3 inches in diameter, at the Franklin, New Jersey, zinc mines. By careful hand picking the value of ore has been increased 26 per cent.—a matter

of considerable importance, since, if ore transportation is \$2 per ton, \$2 will be saved on every 4 tons shipped. At the mines of the Terreira Gold Mining Company, in South Africa, the value of the ore as it comes from the mine is \$17.17. After assortment, it is worth \$25.78, the increased value due to sorting being \$8.61. The increase in value leaves but 84 cents per ton in the waste rock, although the latter is 36 per cent. of the ore mined. The saving here is not only in the increase of values, but in the stamp mill, for the wear and tear on the machinery diminishes in proportion to the reduction in the quantity of barren material stamped.

CONSTRUCTION OF BUILDINGS

29. Building Specifications.—The planning and supervision of building construction are among the duties of the mining engineer. Building plans and specifications are necessary if the cost of construction is to be estimated; and this having been determined, the question arises whether the mine will be benefited by them to an extent sufficient to insure the recovery of the cost with interest. To draw up specifications, one must know the value of the building material delivered, the cost of labor, masonry, and machinery, and also the time necessary for construction. The best way to estimate the cost of buildings is by the cubic foot; if a building of known dimensions in a certain locality costs a certain amount per cubic foot, that basis may be taken for the cost of a similar building, either larger or smaller.

In estimating the cost of buildings, the labor should not exceed 60 per cent. of the cost of materials. This rule will apply to any district, whether the labor there is cheap or costly, since materials will correspond in price.

30. Masonry.—The cost of masonry is particularly difficult to estimate in a new country, for at times, even with good stone quarries near by, it may be economy to import stone or bricks from a distance. Masonry should not exceed

$10 per cubic yard, and from that price it can be made to taper down to $1.50. In some instances, stone can be quarried and delivered to masons for 75 cents per cubic yard, and when such favorable circumstances exist masonry should not cost more than $1.50. But this is exceptional. The usual figure for good cement-mortar wall, pointed and well bound, is about $2.50 per cubic yard. The stones for mine buildings are not usually dressed, but are faced and split by the mason that lays them.

31. **Foundation Walls.**—Masonry at mines is confined usually to foundation walls and engine beds, although sometimes it may have a rather wider scope. A good rule to follow when constructing foundation walls for heavy buildings is to give the wall plenty of binding material, composed of two-thirds cement and one-third sharp quartz sand, and to have all spaces filled with spalls and thoroughly grouted. Headers and corner stones should be properly laid to strengthen the bond, and course joints should be properly broken. The batter generally given to such walls varies from $\frac{1}{2}$ inch to $1\frac{1}{2}$ inches per foot, from the surface up, and while foundation walls should go to bed rock, it is not necessary to batter them below the surface, unless it is desired to give them an extra wide base. A firm foundation wall whose top is less than 20 inches wide cannot readily be constructed of rough stone, especially if it must carry heavy structures subject to more or less vibration, as mine buildings must necessarily be. When the structures are placed on side hills, the foundation walls answer also as retaining walls to keep the earth from sliding.

32. **Retaining Walls.**—The usual form of a retaining wall is shown in Fig. 27. There is no fixed rule for determining the dimensions of retaining walls, but the one given below will probably meet every requirement.

Rule.—*When the backing is loose, a wall of first-class large stones laid in mortar should have a base CD equal to one-third its vertical height DB. A wall of bricks laid in mortar should have a base of two-fifths its vertical height.*

Retaining walls should have firm, wide foundations. These are made wider at the base than the walls themselves and are continued up to the ground level, as shown at CG. The retaining wall is built at a distance GF back from the face of the foundation and given a batter of about $1\frac{1}{2}$ inches to the foot from the base to the top. The object of this construction is to allow the rain water to drain away from the foundation. In case the walls are built against rock, they

FIG. 27

should be well backed with loose stones; and in those instances where the rocks carry water, all spaces should be filled with cement to hold back the water, or a drain should be made to carry the water away from the masonry. The latter plan will probably be better when the wall is at the foot of a hill, or where the pressure against the wall is likely to be considerable. If there are other channels for the water to escape—a matter easily determined by an examination of the strata—it will not make much difference if the backing is not water-tight.

33. Cement.—Concrete walls are widely used for foundations, retaining walls, and other purposes, where it has been the custom in former years to use stone or brick masonry. The proportion of cement to broken stone and sand in concrete varies according to the strength demanded by the engineer. A strong retaining wall may be constructed

of concrete composed of broken stone that will pass through rings 1½ inches in diameter and over 1-inch rings, 2 parts of clean sharp quartz sand, and 1 part of hydraulic cement.

For especially strong walls, use smaller stones and more cement; for example, 3 parts broken stone ¾ inch in diameter, 2 parts quartz sand, and 1 part cement. The stone should be quartzite, granite, good sandstone, or some other kind of stone largely composed of quartz, since some chemical action takes place between quartz and cement.

Two kinds of hydraulic cement are used, one known as *natural-rock cement* and the other as *Portland cement*. Natural-rock cement contains silica, lime, magnesia, iron oxide, and alumina in such proportions that, when burned, slagged, and ground fine, it will produce a cement that will harden under water. The natural cement most in use is known as *Rosendale cement*, and comes from a place near Rondout, New York, on the Hudson River.

Portland cement is an artificial product composed of materials proportioned to furnish a slag that, when ground fine, produces a more uniform hydraulic cement than the natural-rock cement. It appears from experiments that the finer the cement is pulverized the better it becomes.

Hydraulic mortar is composed of 1 part clean sharp sand and 2 parts Portland cement.

Cement mortar is composed of 3 parts sand and 1 part Portland cement.

34. Concrete.—*One-three-six* is the general term given to concrete for mine portals, tunnel linings, and foundations; it consists of 1 part of cement, 3 parts of sharp silica sand, and 6 parts of broken stone from ½ inch to 1½ inches in diameter. These materials are to be thoroughly mixed with sufficient water to make a thin composition, and then rammed into place in layers until the mud accumulates on the upper surface of the concrete. Boxes, centers, and forms of the size of the desired wall are constructed, and into these the green concrete is dumped and rammed in courses. After the concrete has set or hardened, the boxes are removed

and the wall is complete. If the material has not been thoroughly tamped so as to force the sand and cement into all the spaces, cavities will show in the concrete at the sides, and therefore similar cavities may be expected in the wall as well. In order to avoid this and allow the air to escape, a spade should be thrust into the concrete and worked until all the air is out; this should be done after the first and before the second tamping.

35. Guarding Against Frost.—In places where freezing is likely to occur, the back of the wall should be sloped, as shown at *a b*, Fig. 28, and smoothly finished to lessen the hold of the frost, which otherwise might displace the masonry. The foot of the slope *b* should be at the frost line, usually about 2 feet below the surface in moderate climates and from 4 to 6 feet in climates as far north as the 45th parallel of latitude. High altitudes will also affect the frost line.

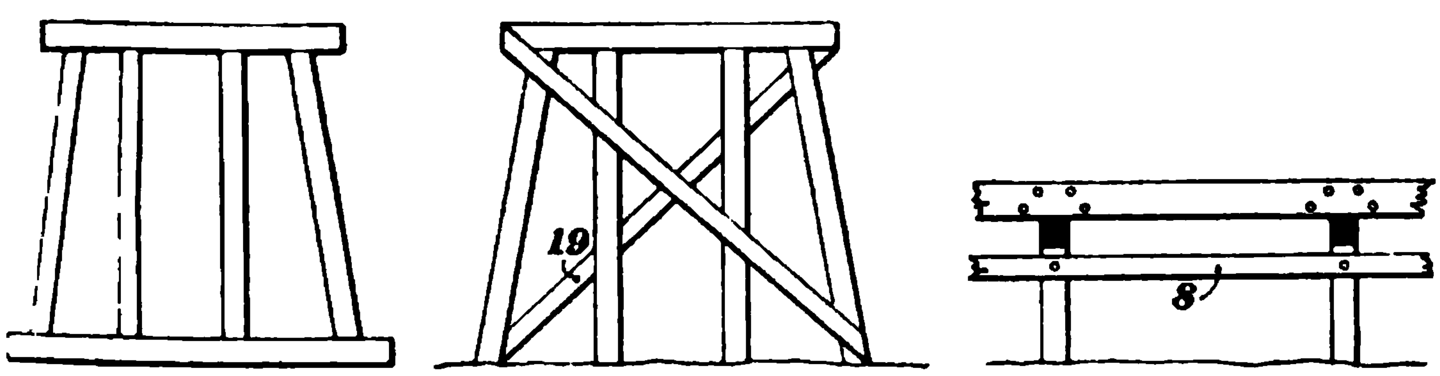
Fig. 28

36. Stability of Retaining Walls.—Having proportioned a retaining wall by the rule given in Art. **32**, its stability may be increased by stepping it, as shown in Fig. 29, without adding to the volume of the masonry. The offsets are determined as follows: Through *e*, the middle point of the back, draw any line *f g*. From *f* erect the perpendicular *f h*; divide *g h* into any even number of parts, in this instance four, and through these points draw division lines parallel to *f h*. Next, divide *f h* into one greater number of equal parts than *g h*, and through these points of division draw lines at right angles to *f h*, forming the offsets shown in the figure. By increasing the thickness of the wall at the base, the center of gravity is lowered and the stability consequently increased. The backing included between the lines *g h* and *f h* exerts only vertical pressure against the offsets, which tends greatly to prevent the overturning of the wall.

Fig. 29

FRAMING TIMBER STRUCTURES

37. Trestles.—As most mines require railroad tracks, and as it is frequently necessary to construct trestles, leading either to bins or to dumps, during the regular construction of any mine railroad, illustrations are given for forming some simple forms of trestles. Fig. 30 (*a*) shows a trestle bent complete. The lower member is the sill, the two side members are the braces, the two center pieces are the posts or legs, and the top horizontal timber is the cap. Fig. 30 (*b*) shows another bent with cross-braces (numbered *19*); these are used on pile bents or bents having long posts. Fig. 30 (*c*) shows the cap and legs of two bents surmounted by a stringer, and the legs of the two bents joined by a tie-piece *8*.



(*a*) (*b*) (*c*)

FIG. 30

Fig. 31 shows the various parts of a trestle: *3* is the cap piece notched at *5* for the brace and notched and mortised at *9* for the leg; and *4* is a cross-tie notched for the four stringers *18* that rest on the cap piece. It will be noticed that two of the stringers are notched to cover the cap, and that two are notched so as to cover only half the cap. In the same way, the jack-stringers *7* are notched, at *5*, the object being to break joints. The rail guard *6* is notched to fit over the cross-ties *4* and prevent a car, if it should happen to jump the track, from going off the trestle. The packing-block *11* is fastened to the stringer *18* by packing-bolts or ship's spikes *12*, with the object of keeping two adjacent stringers in line, as shown in Fig. 39. *13* is a round brace and *14* a round leg for a pile bent. *15* is a squared leg tenoned at *20* for the mortise *9* in the cap, and also bored for a treenail. *16* is a brace tenoned at *20* and mitered to fit the dap *5* in the

cap. *17* is a sill, resting on a mud-sill *10* to keep it from the ground and prevent it from decaying. Sills are dapped as at *5* for the brace *16* and generally mortised for the leg; the mortise is not shown in the figure, but is similar to *9*, treenails being used as in the caps.

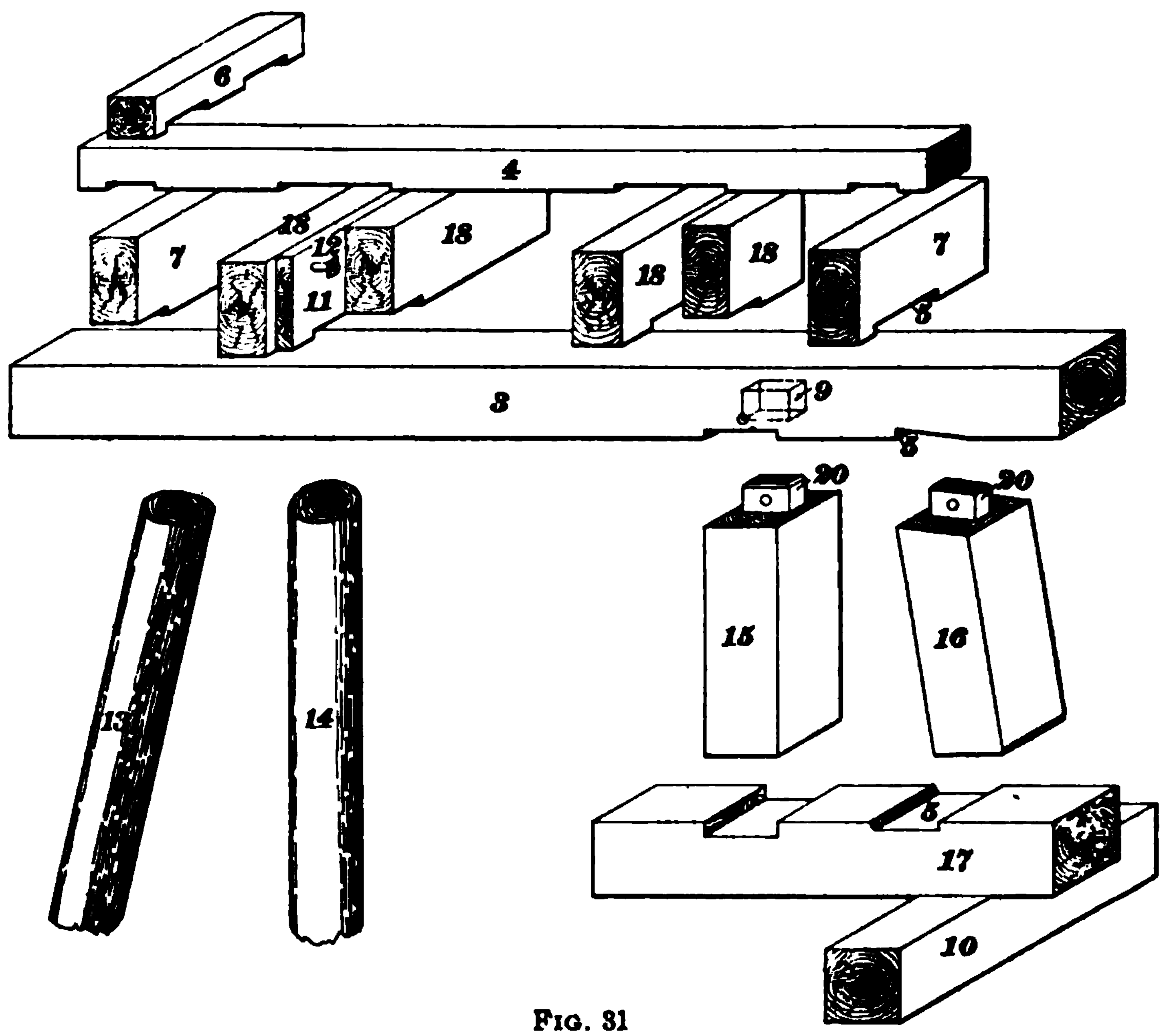


FIG. 31

38. Pile Trestles.—Fig. 30 (*b*) is a trestle constructed of piles driven into the earth. Fig. 31 shows the angles at which the piles *13* and *14* are driven to form the legs and the braces. In case the trestle is not a high one, the piles may be driven straight, but batter braces are considered better in every case.

The caps for pile bents are usually of sawed timber, and the piles are tenoned at the upper end, as shown in Fig. 32 (*a*), to fit mortises in the cap. When this method of securing caps is used, a hole is drilled for a treenail through the cheeks of the mortise in the cap and through the tenon. It is well to have the holes in the cheeks of the mortise so

placed that when the pin is driven through two holes it will tend to draw the cap down on to the top of the pile. The pin used for this purpose is called a *treenail*, and should be made of hard wood, such as locust or hickory, if possible, and slightly tapered, as shown in Fig. 32 (*b*). In most mining districts of the western parts of the United States, the hard woods mentioned must be shipped from distant localities at a cost that is practically prohibitive; consequently, it is customary to use for treenails the strongest and most enduring wood to be found in the locality. In Colorado, the preference is given to red hemlock or red pine, rather than to fir and white pine, for both treenails and heavy construction work. Sometimes the caps are not mortised and tenoned on the piles, but are secured by means of drift bolts, as shown in Fig. 33, or by means of dowels, as shown in Fig. 34.

(*b*)

FIG. 32

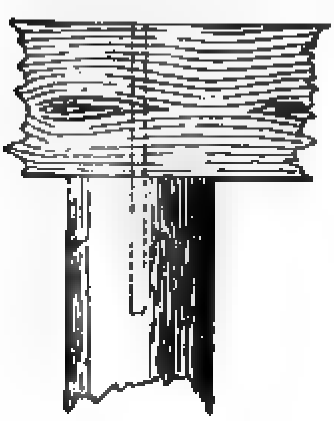


FIG. 33

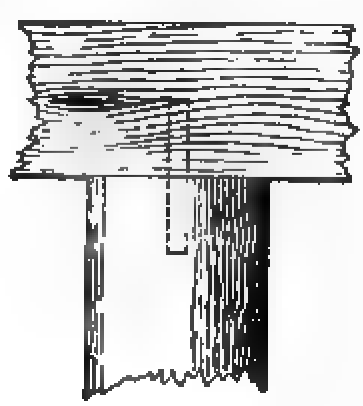


FIG. 34

39. Split Caps.—Another arrangement for pile-bent caps is shown in Fig. 35. This is called the *split cap*, as, in place of using one 10″ × 10″ timber, two 4″ × 10″ timbers are employed, and the top of the post is cut as shown in the figure. The timbers can be seen at *a* and *b*, while *c* is a tenon, the full width of the post, that is allowed to project up between the timbers. No notches are cut in the timbers where they rest on the tops of the posts; they are secured in place by means of a bolt *d* that passes through both timbers and the tenon. This framing can be used with either squared or round posts, and, on account of

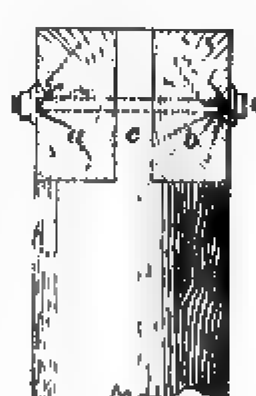


FIG. 35

the smaller size of the cap pieces, it is possible, in some cases, to obtain better timber. Repairs, also, can be made with greater ease than where caps are mortised and tenoned or fastened with drift bolts to the tops of the posts, for either of the caps can be removed and replaced without interfering with traffic, and without cutting any portion of the timber work.

40. Framed Bents.—Where it is not necessary to drive piles to form pile bents, framed bents are used. Fig. 36 shows a framed bent in which all the timbers are secured by means of drift bolts.

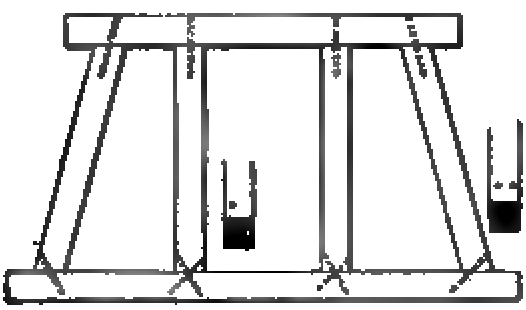


Fig. 36

41. Foundations for Sills. The sill of the bent should always be placed on some form of foundation. This may be composed of timber mud-sills, as shown at *10*, Fig. 31, but it is better practice to construct stone or masonry walls under the sills and to see that the latter are well bedded. When masonry is used as a foundation for sills, care should be taken to see that the stones are well laid; it is never good practice to construct these foundations of rounded stones laid up like rubblework, for the constant passage of trains over the trestle is liable to break up such a foundation. The use of round stones for any masonry work on which pressure will fall is to be avoided. If regular masonry is out of the question, concrete may be used to advantage.

42. Placing Timbers.—The batter braces for trestles should have a uniform angle of 3 inches per foot. Fig. 37 illustrates the method of framing at the foot of the batter braces and posts in

Fig. 37

framed bents, and also shows a drainage hole bored in such a manner that any water collecting under the jointing will immediately flow out through the drain and thus reduce the tendency that timbers have to rot. It is well to remember,

in connection with trestlework, that green oak timbers or wet oak timbers spiked or bolted with iron soon decay about the iron. Fig. 38 shows the method of mortising and tenoning the legs to sills.

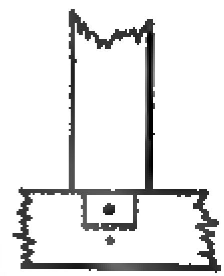


FIG. 38

43. Illustration of a Framed Bent. Fig. 39 is a dimensioned drawing showing a timber bent as used on one line of railroad. The gauge of the track is standard, that is, 4 feet 8½ inches, and the dimensions on the drawing fully explain the various parts. On the right, the bent rests on piles, while on the left it rests on rock foundations.

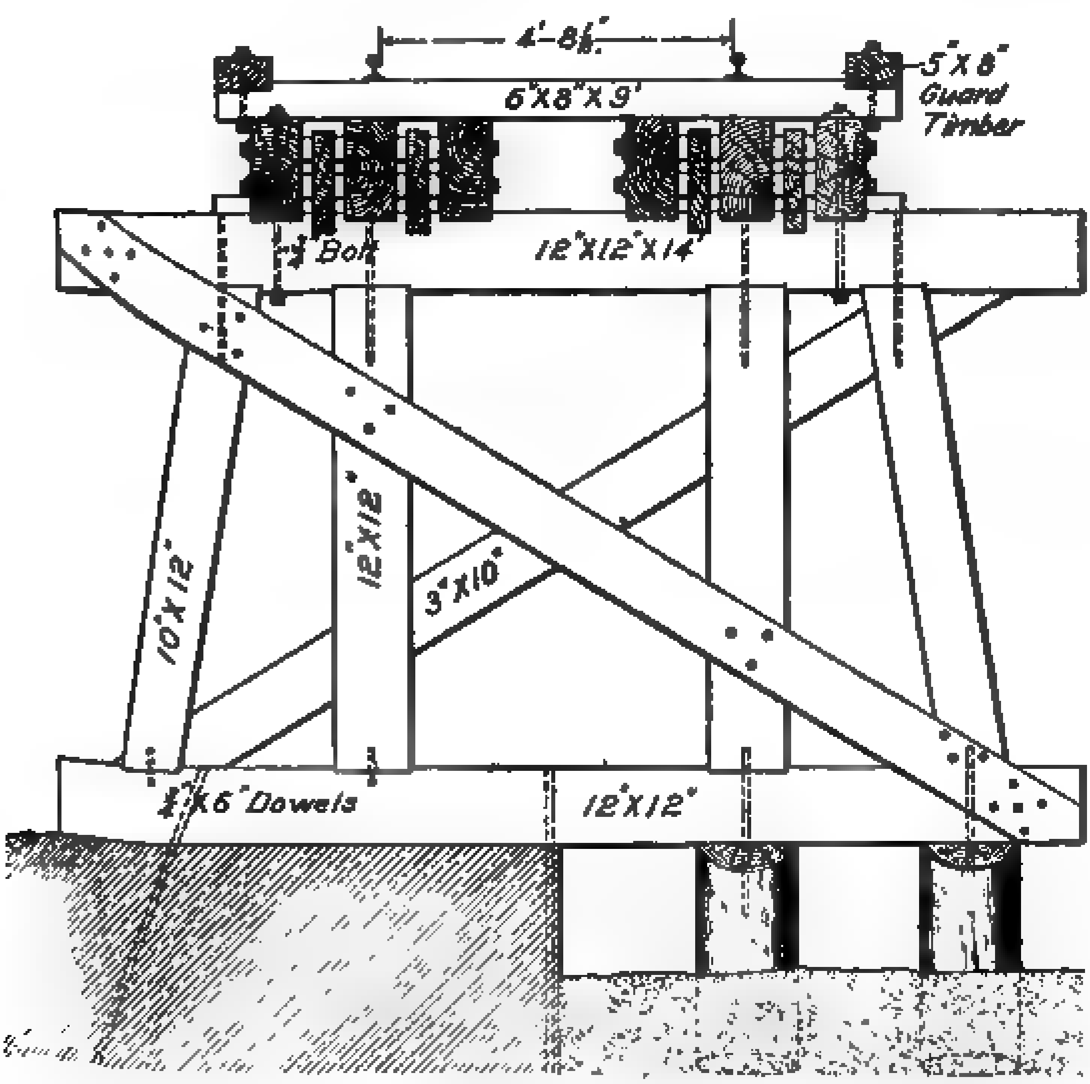


FIG 39

44. Elevation of Outer Rail.—Where the trestle comes at a curve in a railroad track, if it is intended that the cars should move over it at any considerable speed, it is necessary

that the outer rail shall be elevated. This may be accomplished by wedge blocks placed on the top of the cap and under the stringers; usually, however, the cross-ties are cut wedge-shaped to give the desired elevation, as this plan makes a firmer track.

45. Ideal Trestle Construction.—Fig. 40 shows an end and a side elevation of a trestle. The stringers *a* and the braces *b* are of Georgia pine; the caps *c* and sills *d* are of white oak; the posts *e* are of red cypress. This combination of timbers is for the purpose of economy; white oak is more expensive than yellow pine, but it is better able to resist

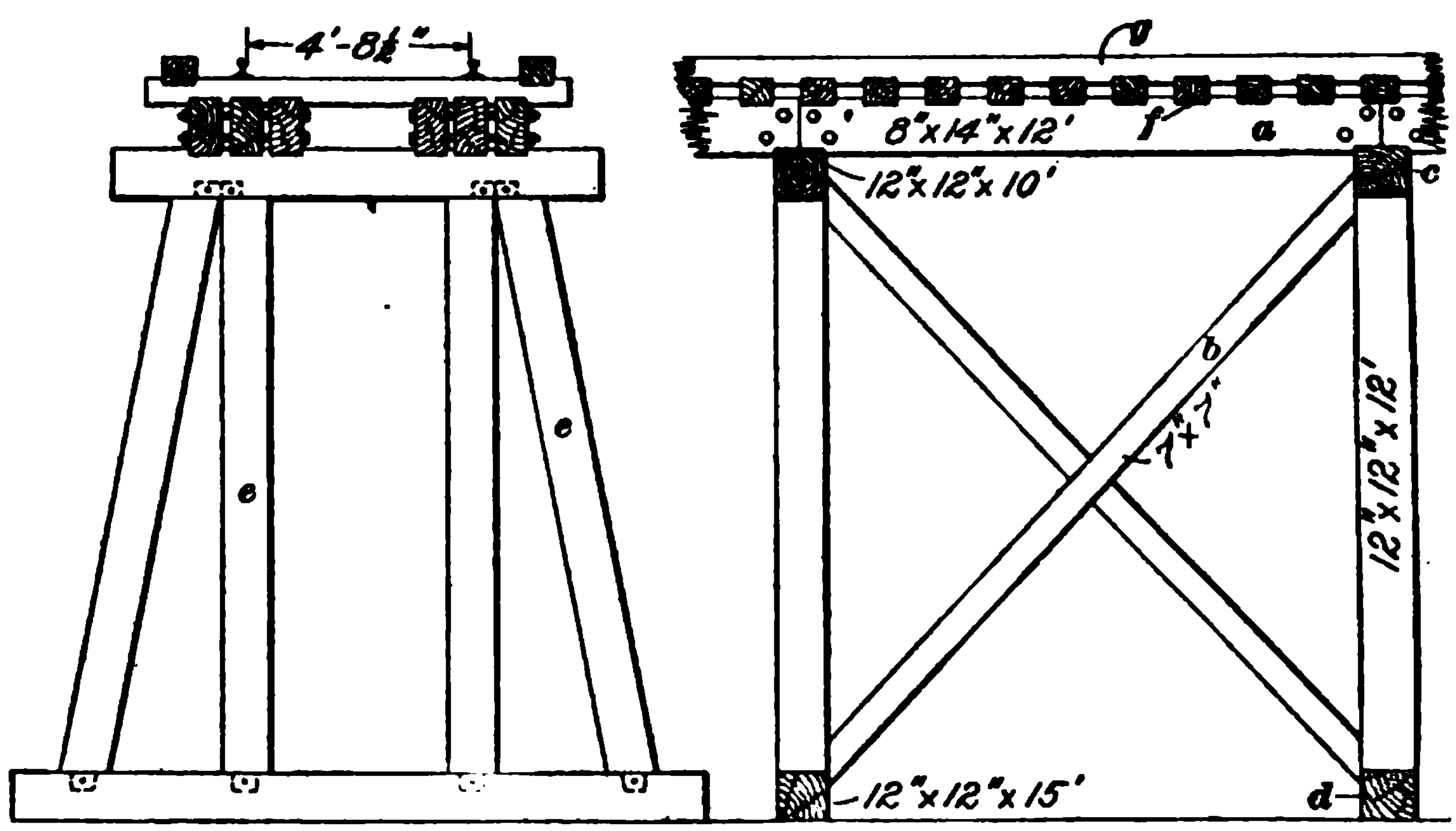


FIG. 40

pressure across the fibers. Stringers of yellow pine can be obtained more readily than oak, and resist endwise pressure better, besides having a greater modulus of elasticity. The cypress posts are durable and possess sufficient strength for such purposes. The stringers, being the most important part of the structure, should be deepened and increased in number if the bents are to be more than 12 feet apart, and if heavy trains are to pass over them. Because of the braces *b*, the legs are mortised and tenoned into cap and sill. The guard-rails *g* are notched to fit the cross-ties; and to prevent any chance of the cross-ties *f* spreading, every other tie is bolted to the guard-rail.

ORE BINS

46. Construction of Ore Bins.—Ore bins have a twofold object: they afford a place for the storage of ore until it can be shipped, and they furnish a place where ore can be sorted and picked over in daylight as it comes from the mine. Heavy, squared timbers are used in the construction of ore bins, in order to secure rigidity and strength. The strength of the framing timbers is not calculated, but they are taken of the largest convenient size to be had, usually 10 in. × 10 in., or 12 in. × 12 in.

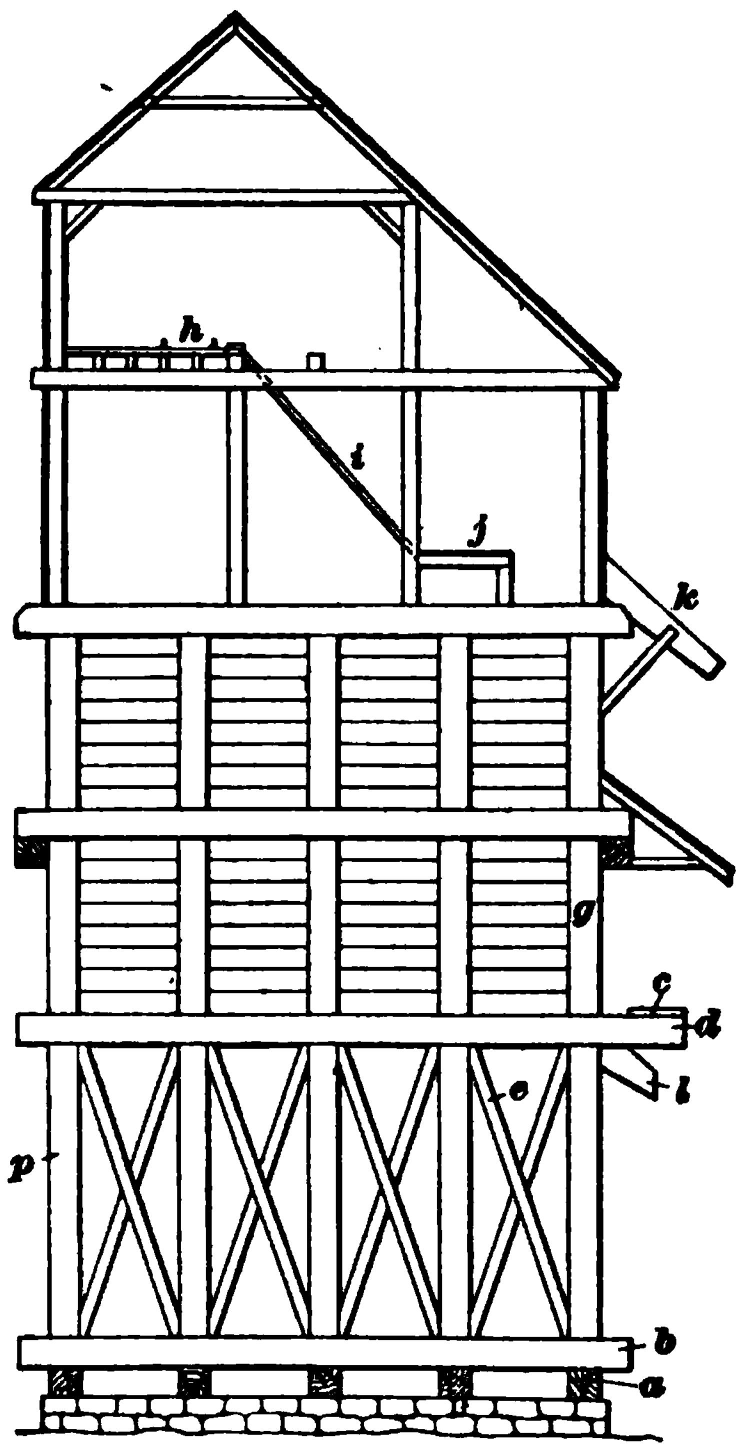


Fig. 41

Fig. 41 shows the end elevation of a modern ore bin. The mud-sills *a*, sills *b*, posts *p*, caps *d*, and, in fact, all the frame timbers between the ground and the sorting floor are of 12″ × 12″ sticks. The posts *p* and *g* are tenoned and pinned to their sills and caps. The braces *e* are of 6″ × 12″ timbers spiked or bolted together where they cross. As these braces are merely stiffeners, the posts being sufficient to support the load, it would be more economical and equally effective to use two 3″ × 12″ timbers, as shown at *a* in Fig. 42. These timbers or planks are placed in a shallow notch made in the post; but even this small amount of cutting can be dispensed with and bolts used for the purpose of keeping them in place.

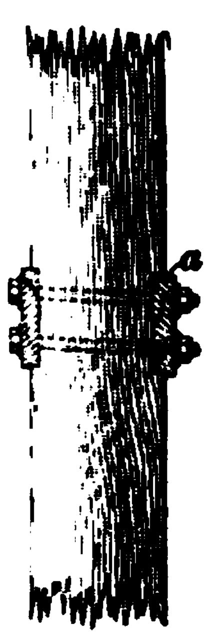


Fig. 42

The framing at the top of the structure is not made so heavy as that below the picking floor, since it has only the roof to support. The cars run in on the track *h*, Fig. 41, and dump the ore over the grizzly *i* on to the picking table *j*, where any pieces of rock are picked out and thrown into the rock chute *k* leading to a rock car (not shown). The chute *l* is for ore that is to be shipped in cars. The men who load the cars with ore stand on the

Fig. 43

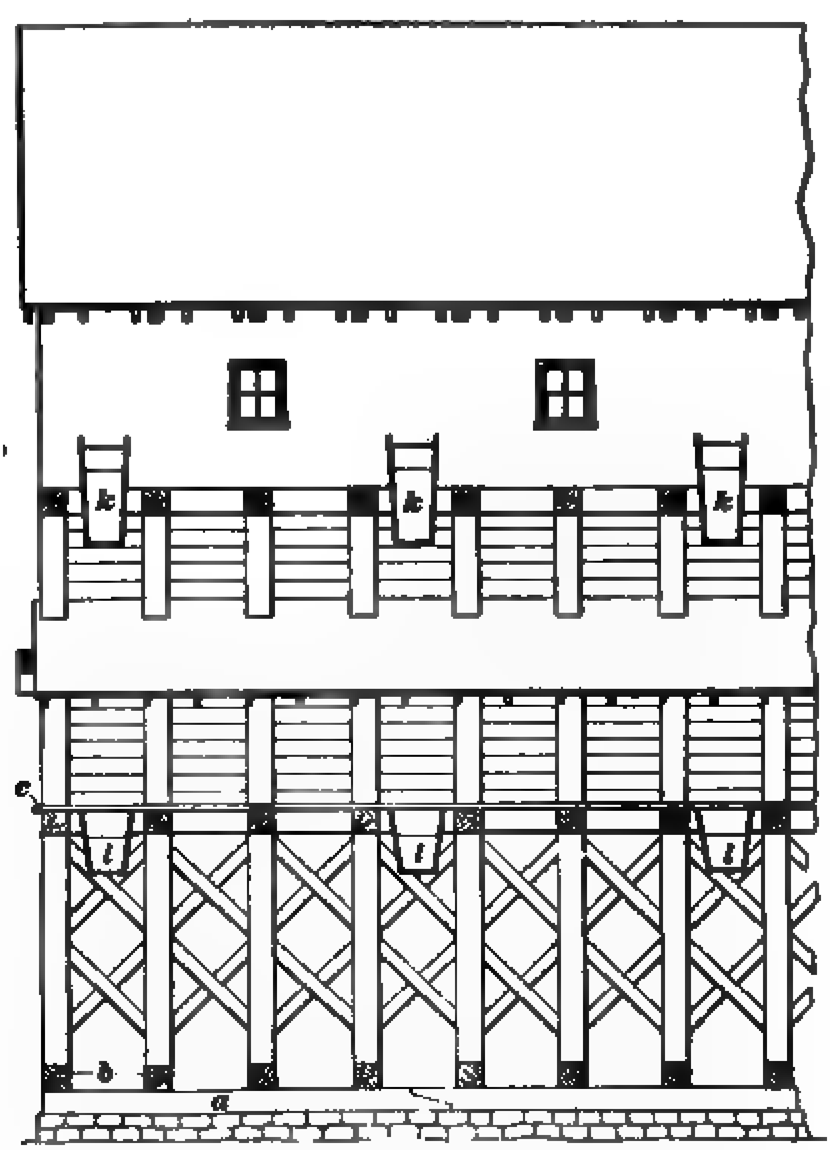


Fig. 44

platform *c* and open and close the ore gate leading to chute *l*.

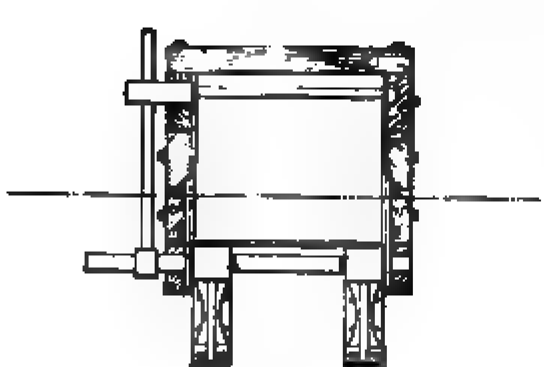
Fig. 43 is a plan of the top of the building, showing the track *h*, the grizzly *i*, the picking table *j*, and the rock chutes *k*.

Fig. 44 is a front elevation of the same ore bin. The mud-sills *a* are seen on the foundation walls, with the bent sills *b* resting on them. If the mud-sills rest on masonry walls, as shown, five such walls will be required running the entire length of the building. Piers, arranged to come under the place where the sills *b* rest on *a*, would probably answer every purpose, and greatly reduce the cost of the masonry. Bracing the posts lengthwise of the building has the same object in view as bracing them across the building, that is, rigidity. *k*, *l*, and *c* are the rock chute, ore chute, and ore platform, respectively.

47. Ore Gates.—Cars may be loaded over the side or lengthwise, the former arrangement being termed *side loading*, and the latter *center loading;* in either case, the ore gate may have the same construction. Ore stored in bins is very apt to pack, making it difficult to open and close an ore gate. To overcome this difficulty, numerous gates have been devised, the most common one of which is known as the *rack-and-pinion gate*, shown in Fig. 45. This will be found serviceable in almost every case where mixed coarse and fine ore is run, and is generally used in the western parts of the United States. Where there is much pressure against the gate, due to a large quantity of coarse ore, the leverage obtained by the hand wheel is not sufficient to force the ore under the gate, either in or out, when it is desired to close it. This feature is objectionable, since a car may be overloaded before the gate can be closed. To overcome this difficulty, the gate shown in Fig. 46 was devised. In this case,

FIG. 45

the gate *a*, at (*a*), is curved; it turns on an axle, as shown, and is moved by a lever *b*. When the lever is pushed back, as in (*a*), the ore slides down the chute over a straight face; but when the lever is pulled forwards, as in (*b*), the curve crowds the ore against *c*, the top of the chute. There is a front to the chute which keeps the ore from running out except when the curved gate permits it. The curved form of this gate permits the ore to be thrust back without difficulty, shutting off the stream of ore whenever it is necessary. At (*c*) is given a front view of this gate, which is attached to the end of a spout.



(c)

Fig. 46

Fig. 47 (*a*) gives the front elevation of an ore bin, with the gate *a*, lever *b*, platform *c* (from which a man works the lever), and the chute *d*. A side sectional elevation of the gate is shown in Fig. 47 (*b*), in which the reference letters have the same significance. In the gate shown in Fig. 46, the lever is moved toward the bin when it is desired to open the chute; but in the gate shown in Fig. 47 the lever is pulled away from the bin in order to open the chute, and toward the ore bin when the flow of ore is to be shut off. Any ore that might obstruct the gates while being closed will be pushed down the chute, and not back into the ore bin.

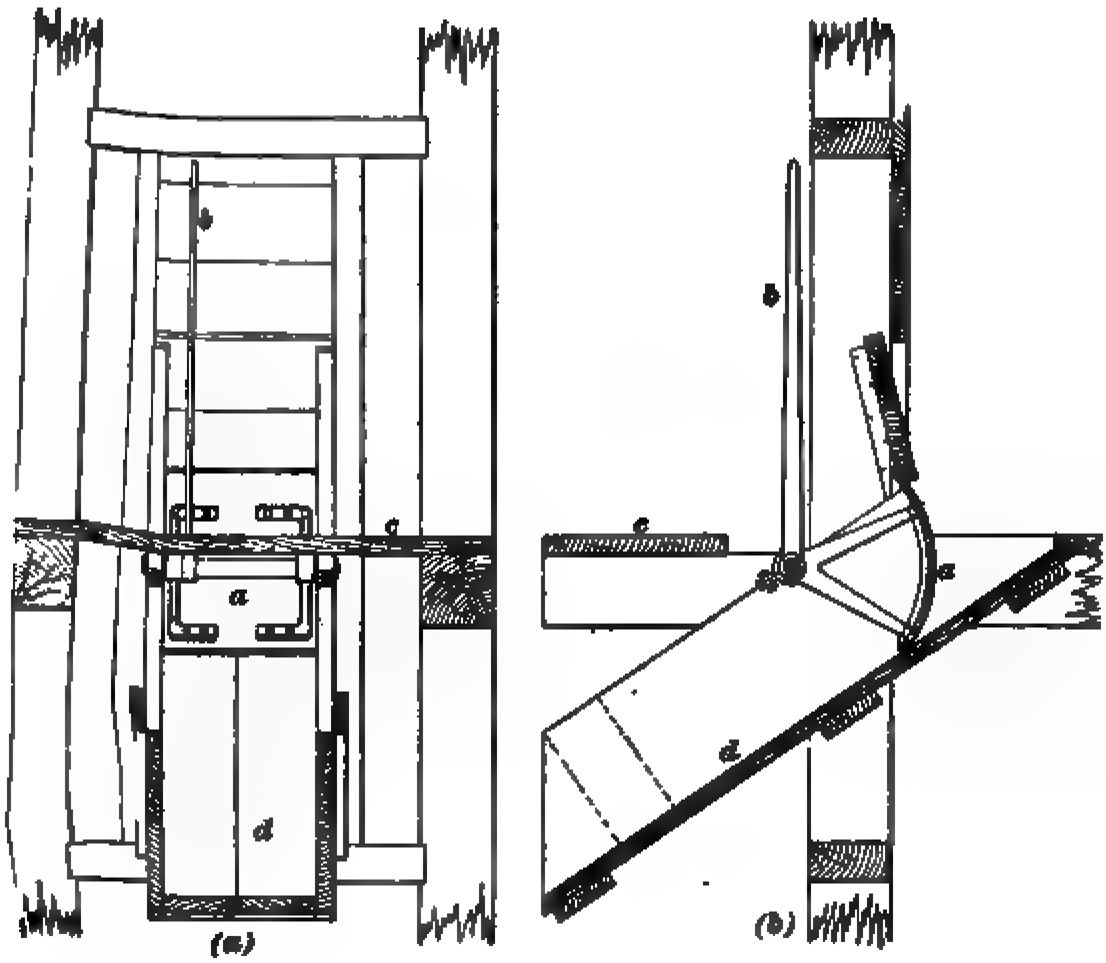


Fig. 47

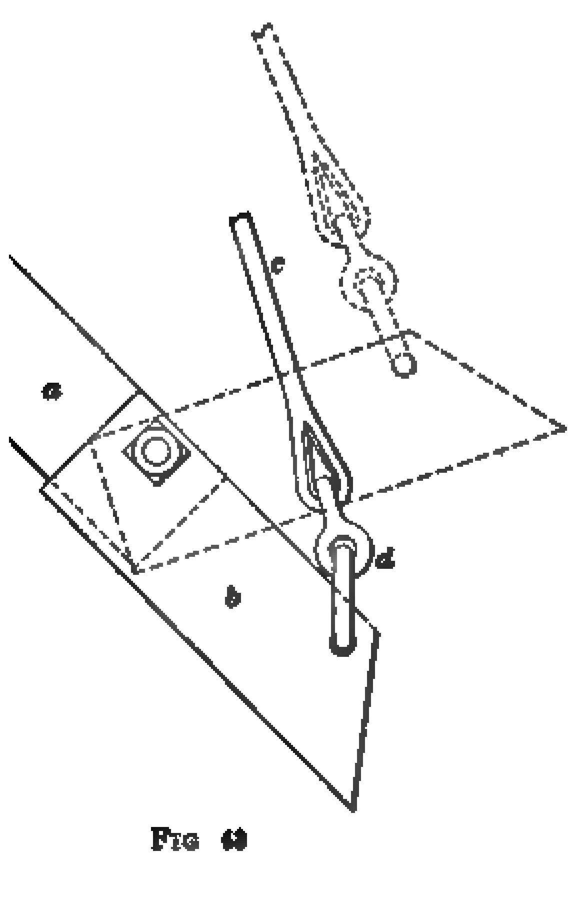


Fig. 48

48. Swinging Chute Gate.—Where the opening from an ore bin is not large and the ore is comparatively coarse, a movable apron as at *b*, attached to a chute *a*, Fig. 48, will shut off the flow of ore effectually. The apron is raised to the position shown by the dotted lines by means of a rope *c* attached to a bail *d*; when the apron is in this position the ore is kept from running out of the chute. But when fine ore is mixed with the coarse, the fine works out between the joint made by *a* and *b*, thus causing trouble and sometimes preventing the apron from being lowered to the position *b*.

Swinging aprons are frequently used with other gates, as by their use it is possible to load cars to better advantage and with less trimming; that is, it is possible to pile the ore evenly over the trucks of the car.

Fig. 49

49. Sectional Gate for Coarse Ore. Fig. 49 illustrates a pocket stop for use with coarse ore. In this case, the gate is composed of iron bars *h*, which form a grating to hold

the ore back. To open the gate, an air cylinder *a* is provided. In this cylinder, there is a piston connected with the piston rod *b* and the crosshead *c*, which works between the guides *d*, *d*, and is connected with the iron bars of the grating *h* by means of chains, as shown. The operator stands on the platform *i*, which is placed over and in front of the chute to the cars. When he wishes to raise the bars *h*, he opens the valve *e*, thus introducing compressed air from the pipe *f* into the cylinder *a*. This air raises the piston, and thus the crosshead and gate bars. When he wishes to lower the bars again, he closes the valve *e* and allows the air to escape from under the piston by opening the valve *g*, when the crosshead and bars come down of their own weight. If any of the bars stick, the clevis at the top of the bar can be turned down and the bar driven with a hammer. This form of gate possesses considerable advantage when dealing with coarse ore; and the fact that it can be opened and closed so quickly, with the aid of compressed air, enables the cars to be loaded in less time than otherwise would be the case.

50. Ore-Bin Floors.—In order that the ore may run through the gates by gravity, the floor to ore bins must have an angle of at least 32° if lined with boards, and not less than 28° if lined with iron.

The floor stringers are given centers of 2 feet and are placed at right angles to the front of the chute. Two-inch planks are nailed across them for flooring. These may be of hemlock, but should be well seasoned to prevent shrinkage. Over this flooring, tar paper or some other thick sheathing paper is placed, and over this, at right angles to the other boards, 1-inch oak boards are nailed. A better plan is to use 1-inch tongued-and-grooved pine plank for floor boards and 1-inch oak or maple boards for floor lining. Fine particles of ore will work under the lining, and to save this the floor should be tight.

51. Floor Plates.—Fig. 50 shows an ore bin with an inclined floor and a series of swinging ore chutes. As a thousand tons of ore go over the floor daily in the shipping

season, even oak or maple linings would wear out quickly; for this reason ½-inch cast-iron plates are used to line the floor. These plates are made in 2-foot squares, and are bolted down, the head of the bolt being received in a countersink in the plate. Some object to bolts and use spikes, claiming that the fine ore will work under the plates and break them; on the other hand, fine ore will work under the plates even when spikes are used, so that, if the spikes

FIG. 50

do not pull out of the wood, the danger of breaking the plates is as great as in the other case. As wet ore will freeze to iron plates, it should not be allowed to remain in the bins during freezing weather.

Crusher and sampling floors also are protected with cast-iron plates; in fact, it is unusual to find hard-wood floors about mines in the Western States, the cost of wooden floors being more in the long run than the first cost of the iron plates. In some cases, cement floors have been introduced with fair success.

52. Crib Storage Bins.—In localities where dressed timber is scarce and expensive, ore bins and storage bins are sometimes constructed from logs obtained near the mines. Such structures are not difficult to make, are serviceable,

and may be quite large; besides, they possess the advantage of being readily increased in height when greater capacity is necessary. If built up with four sides, the timbers holding the sides together should not be more than 12 feet between centers; the same distance will answer when the bin is backed against a hillside. In the latter case, the ties extend from the front to the hill and rest on the earth. At first it may be necessary to hold the hill end of the tie in place by weighting it down; but when once the tie has been covered with ore it will remain in place. The ties and front logs are notched and further held by ship's spikes driven through one into the other. Storage bins of this description are quite general in the mountains of Southwestern Colorado, but are not confined to any one locality. Miners and surface employes usually possess enough mechanical skill to construct bins that are serviceable and symmetrical and in conformity with local conditions.

Crib storage bins are provided with ore gates for convenience in loading, and sometimes with rough floors made of logs dressed on one side. The spaces between the logs are not filled up, as they become stopped with the larger pieces of ore, which, when dumped from the ore cars, roll a greater distance than the smaller pieces.

53. Stone Ore Bins.—In places where timber is not readily obtainable, ore may be confined within roughly piled walls made up with the largest pieces of ore or with barren rock. These walls are given a batter, the same as retaining walls, and are increased in height as the ore accumulates. Such walls are temporary, and are chiefly used for the purpose of storing second-class ore until it can be disposed of profitably. It may be necessary, however, to use such walls for good ore when the shipping season is short.

54. Assorting Ore.—Where the output of a mine is not large and the ore is lean, it is customary to break the richer pieces of ore from the poorer pieces with a hammer. This is termed **cobbing**, and has for its object the reduction of the pieces in size, the removal of worthless rock from the

ore, and the separation of the lean ore from the rich, thus making two or more grades of shipping and milling ore. In the earlier days of mining, ores were sorted into two or three classes to meet graded prices at smelting works; but within recent years, the former scale of prices has been discarded and the ore is paid for at a stated price per ounce of gold or silver, together with the percentage of lead or copper that the ore contains. There are sometimes refractory elements, such as zinc, silica, and arsenic, which are not desirable for smelters, and when these elements exceed a certain designated percentage a proportionate charge is made against the account of the shipper. When ore carries a high percentage of gold and silver it is customary to assort it on the premises and place the most valuable part in ore sacks of special manufacture, in which condition it is shipped to the smelter or mill. Assorting is carried on with the purpose of throwing out ore containing refractory elements in excessive quantities, so that all the ore shipped will yield a net profit. Ore of lower value than shipping ore is stored until some appropriate system of treatment can be applied. With mines at high altitudes, from which ore cannot be shipped during the winter, it is customary to assort and store the ore in convenient places, either inside or outside the mines, until the roads or burro trails will permit its shipment.

At times lean ore can be worked up into shipping ore by hand, a small hammer being used to knock off the mineral; this is termed **bucking**. Where the output is large, quicker means must be adopted, such as rock crushers for breaking the ore, and picking belts or picking tables on which the ore is assorted.

The object of picking ore is to separate the output of the mine into two or more classes: for example, (1) the shipping ore—ore with a percentage of value high enough to warrant its being delivered immediately to the smelter; (2) the ore to be subjected to lixiviation, or to be worked by chlorination, cyaniding, or some other wet process; (3) concentrating ore.

55. Some mines with a large daily tonnage have their own ore-treating plants, where amalgamation, concentration, and lixiviation are carried on, thus converting the products into bullion and thereby saving smelter, freight, and custom mill charges. Mines that have their own smelting plants need not be so particular in their assortment of ore.

Again, the object of picking may be to convert ore that is objectionable to the smelters or lixiviation mills into a product that may be treated by them. The fine ore is rarely passed on to the picking table, though sometimes, where a special picking table is employed, the entire product is passed over it. This is especially true when tables of the bumping pattern are employed, in which the ore is forced along by the bumping or jerking action of the table itself.

56. Picking is usually done by hand, though sometimes a fork is employed, by means of which large masses of barren material requiring recrushing are thrown out from the ore as it is being handled on the picking floors. In cases where the material being assorted contains rich bunches of mineral in rock, it is sometimes separated on the picking floor by bucking. Where picking tables or belts are employed, the persons that do the picking stand at the sides of the table and pick off the different classes. It is best to pick off the larger class—that is, if there is more ore than rock, to pick off the barren rock and allow the ore to pass on over the table. If the larger class were picked off it would entail a great amount of work. Where two or more minerals are being picked from the same belt or table, different persons pick for the different minerals; in this way purer material can be obtained, for each person's eyes become accustomed to looking for a certain product, and they become more expert in picking out one grade than they would if two or three grades were picked out by each.

57. At most gold or silver mines, particularly those in which the ore is found as a sulphide, or a telluride, etc., the fine ore is often the richest. This cannot be assorted by hand; but it should not be thrown on the rock piles; it should

be assayed and shipped with coarser ore. The fine ore resulting from cobbing or bucking should also be saved, assayed, and shipped with coarser ore, provided the values warrant it. The object of assaying this fine ore is to ascertain its value. If it is very rich, it may be added to a shipment of second-class ore in order to raise the value of the shipment; if it is only of medium value, it may be added to a shipment of richer ore.

Frequently, miners are under the impression that their ore is richer than it really is; therefore, some independent assayer should be given a sample for assay. If this can be done before shipments are made the miner will know approximately what he should receive for his ore. If the ore will not pay for its freight and other charges, he can reassort it to bring up its value. In assorting lean ore, care should be taken to see that the good ore obtained will pay twice as much as the cost of the labor expended in its assortment.

Fig. 51

58. Stock Piles.—Where the ore is shipped during a portion of the year only, as, for instance, at the Lake Superior iron mines, from which ore is shipped during the summer months, it becomes necessary to stack the ore mined during the winter in large stock piles. In Fig. 50 a stock

pile *a* can be seen beyond the pockets, and Fig. 51 is a view taken from the opposite direction, showing the pockets and the stock piles on the sides of the track. Stock piles usually

FIG 52

have board floors, in order to keep the material from becoming mixed with the underlying earth or rock. Fig. 52 is a view of a stock pile showing the board floor at the end of the pile.

59. Timber Work in Stock Piles.—Where steam shovels load the ore, it is not advisable to have permanent timbering within the stock pile, as, for instance, a trestle; for, when the time comes to load the ore for shipment, the timbers buried in it will interfere with the action of the shovels. On this account many of the stock piles are formed, as shown in Fig. 52, by dumping the ore over the end of the trestle from the shaft house and then advancing the pile by means of a track laid along the top of the pile. Where the ore is to be loaded by hand these objections to timbering do not apply; but even in this case they are not always a necessity, and should be avoided whenever possible on account of their expense.

60. Special Stock-Pile Trestles.—In case the cars are trammed by power to the stock pile, trestlework is provided,

with the bents so designed that the greater portion of the woodwork may be removed after the pile is completed.

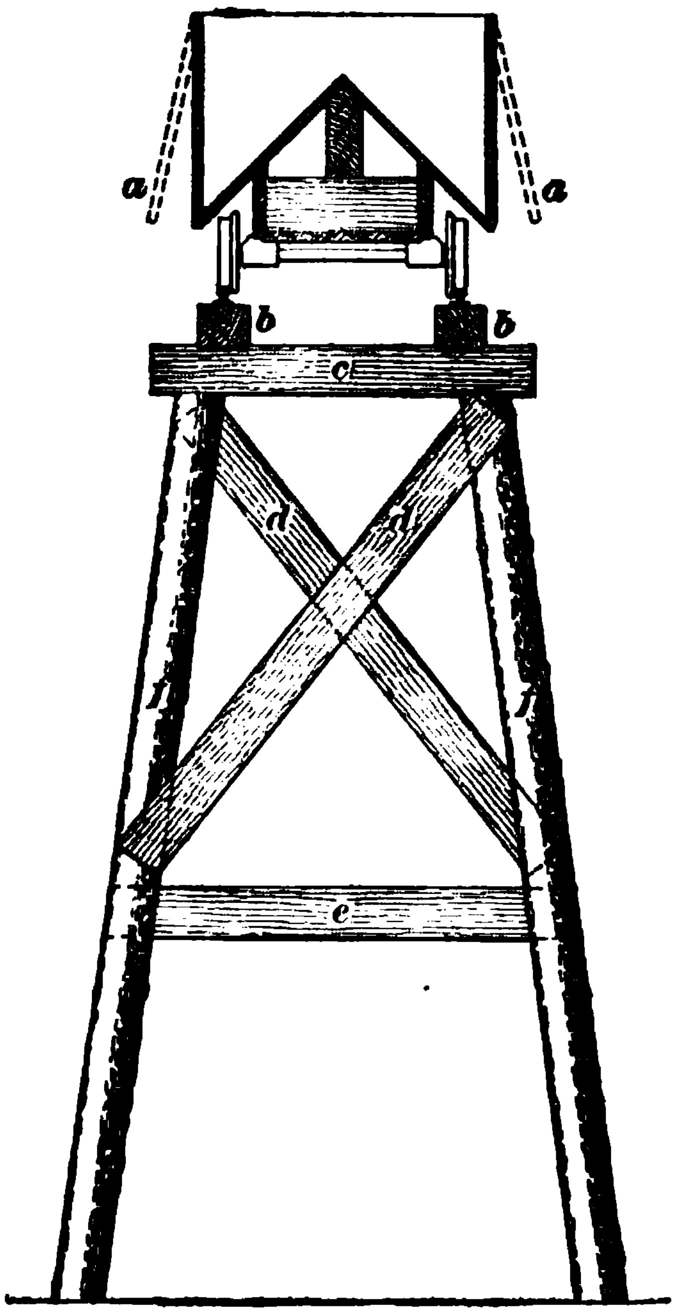


FIG. 53

Fig. 53 illustrates one bent of a trestle designed for stocking ore. The bottoms of the dump cars used in such cases slope downwards from the center to the sides, like an inverted **V**; the doors *a* are kept closed by latches, which are released by pins or blocks placed at suitable points along the trestle. When the doors are released, the ore presses them out into the position shown by the dotted lines and drops on to the stock pile beneath. After the ore has risen around the legs nearly to the brace *e*, the braces *e* and *d* may be removed, as the ore will hold the legs in place. After the stock pile is completed, the stringers *b*, the track, and the caps *c* are removed. This leaves nothing but the legs of the trestle for the steam shovel to pull out.

61. The Hunt Automatic Railway.—Fig. 54 shows a Hunt car, which runs on a narrow-gauge track. It is discharged by means of a tripping block *a*, placed where the load is to be dumped. The sides of the car are fastened to each other in such a way that when one side opens the other opens also. The load is thus evenly discharged and there is no danger of overturning the car. The car bottom slopes from a ridge along the center, so that directly the sides are unfastened the ore runs out. Fig. 55 shows the method of unloading a boat and stacking ore. The bucket *a* is loaded in the vessel and hoisted until it meets a trolley, which then

carries it up the incline to the chute *c*, where it dumps its load automatically. The car *b* is loaded from this chute and is pushed out to the inclined trestle *d*.

Fig. 54

The energy that the loaded car accumulates when descending the incline is stored up in the weight *g*, so that when the

Fig. 55

car is discharged at *f* it is returned to the loading chute. This is accomplished by the car, in its journey down the incline,

picking up a cable that is attached to the weight *g*. The car raises this counterbalance *g* a short distance, but does it gradually, so as to prevent strains on the various parts as far as possible. When the car has dumped, the falling weight gives the car sufficient momentum to carry it up the plane to the chute, after which it is ready to receive another load. This system is extensively used for unloading or for making stock piles; it is used also at smelting plants for ore bedding.

62. Cantilever Cranes.—In another method employed for making stock piles, the **cantilever crane** is used. While this is applied to iron ore and coal, it may be used in other cases. Fig. 56 shows a cantilever used on the Chicago drainage canal; it had a length of 353 feet and a height of 80 feet at the dump. It was mounted on a truck whose wheel base was 57 feet. This truck ran on a portable track, so that the entire structure could be moved with comparative ease. The cars or buckets were first loaded, then raised by rope and a hoisting engine to a trolley, which carried them up the incline to the place from which they were emptied automatically. With such an arrangement a stock pile can be continued indefinitely. An anthracite stock pile containing 250,000 tons was made in this way. Similar in principle are the stationary cantilevers used at ore docks in Cleveland, Ohio, and Buffalo, New York, where large quantities of ore are taken from boats.

63. Loading Ore From Stock Piles.—The ore in the stock piles is loaded into the regular railroad cars during the summer by means of specially designed steam shovels. Owing to the fact that the ore hoisted from the mine always contains more or less moisture, and that the winters in the Lake Superior region are very severe, it has been found that, where a stock pile has been built slowly, so that the layer of ore dumped over the face amounts in a day to only a few inches, the material will freeze solid; and in that cold northern climate this mass will not thaw out for a number of years. The result is that the shovel has to dig frozen ore

Fig. 38

during the summer months. On this account, mines are now concentrating their stock piles at one place only; and by working rapidly and dumping several feet of ore during the day they succeed in preventing the ore from freezing solid, and hence in making it very much easier to shovel.

Where it is necessary to stock ore, provision must be made, in laying out buildings and tracks on the surface, for the convenient handling of the ore, both when adding it to and taking it from the stock pile.

64. Requirements for Stocking Ore.—Extensive pockets, bins, or other provision for storing a supply of ore are, as a rule, not necessary at gold and silver mines having a mill, for in such cases the material is worked up as fast as it is mined; in the case of copper or iron ores, extensive pockets or bins become necessary; and where the shipping season lasts only a portion of the year stock piles are required. At gold and silver mines the pockets or ore bins should hold one week's supply for the mill, so that the mill will not be idle in case of a stoppage at the mine, and the mine need not be idle in case of a stoppage at the mill.

65. Rock or Ore Dumps.—Rock piles or ore piles are sometimes called *dumps;* for instance, miners may refer to the rock dump, the second-grade dump, and the high-grade dump; in this case the second-grade or high-grade dumps would be really stock piles. The object in separating the ore at the dump into different grades is that at some future time means may be available for treating a lower grade of ore than could be treated profitably at the time of mining. This is especially true in the case of gold or silver mines that are shipping smelting ore and saving any ore that may in the future be concentrated.

Frequently, the dumps (either of rock or ore) are in the way of projected railroads. In such cases it will be necessary to excavate a tunnel through them, and to either timber or line it with metal or masonry or cribbing. Metal or masonry linings are preferable on account of the fact that timber is liable to take fire.

66. Dumping Cradle.—At metal mines, when the ore is brought to the surface in mine cars, the cars are usually of the self-dumping type, and no tipple for dumping is required.

Fig. 57

When the ore is hoisted in skips it may be dumped into cars that will transfer it to the pockets, or it may be dumped into cars that are discharged by means of a cradle. Figs. 57 and 58 illustrate one form of cradle and car sometimes

Fig. 58

employed at metal mines. In Fig. 57 the cradle is in the position in which it receives the car. The car is filled with ore from the skips or pockets at the shaft and runs into the cradle over the bins. After the car *a* is in the position shown

in the figure, the end lever *b* is thrown up, dropping the dogs *c* across one of the rails so that the car cannot pass out of the cradle. The upper edges of the car are held in place by the angle irons *d*. After the car is locked in position, the entire cradle is revolved into the position shown in Fig. 58. In most cases the cradle makes a complete revolution, and when it returns to its upright position a dog or latch (not shown in the figure) catches the rails and holds the cradle in a position that leaves the portion of the track on which the car rests alined with the permanent railroad track. After this, the lever *b*, Fig. 57, is thrown down and the car run out on the track. When the car is loaded, the greater part of the weight is above the center of gravity of the cradle, which therefore turns very easily; while after the ore is out the heavy car wheels and track enable the operator to bring it back into its upright position with very little effort.

67. Dumping Mine Cars.—Usually, ore is dumped into bins or on to the stock pile from end or side dumping cars. Figs. 59 and 60 illustrate a common form of mine car employed at metal mines. This car is so constructed that it can be dumped either to the right or to the left, the body

Fig. 59

Fig. 60

being fastened to the trucks by means of a pivot shown at *e*, Fig. 60. When it is desired to dump the car, the operator grasps the handle *c*, and then throws the lever *a* part way over to release the latch *b*, Fig. 59; this, however, does not turn the shaft *d*, Fig. 60, far enough to release the latch that

keeps the front end of the car *f* from swinging open. After this he lifts the handle *c* and swings the car to the right or left, as may be desired; then, on throwing the handle *a* clear over, the front end of the car will swing out and release the contents. After this the car can be swung around again and dropped into place. The front end *f* closes of itself and is secured by returning the lever *a* to the position shown in Fig. 59. The same style of truck is sometimes employed with wooden cars, but, as a rule, steel is the best material of which to manufacture mine cars for use at ore mines, especially at gold and silver mines. A little loop riveted to the side of the car at *g* is for the latch that holds the car in place while it is being hoisted on the cage. When ore is brought to the surface in skips, it is frequently dumped into cars of this pattern before it is run out to the pocket or stock pile.

68. Scoop Cars.—Fig. 61 shows a car having the same character of truck as that shown in Figs. 59 and 60, but provided with a scoop body or box. This style of car can be dumped without unfastening the front, as must be done with the car just shown.

69. Stocking Ore in the Mountains.—In the mining camps throughout Western America, ore bins are sometimes quite elaborate structures; at other times crude. A common method of construction is to erect posts at intervals and join them by horizontal timbers of a similar size. The joints are made by dapping the posts into the sills and caps, or by mortise and tenon. The frame is lined inside with 2-inch or 3-inch planks running horizontally from post to post, and these in turn are reinforced by 1½-inch

FIG. 61

planks that extend vertically from sill to cap. To prevent the opposite posts from bulging, sills and caps are held firmly together by long $1\frac{1}{2}$-inch iron rods that are threaded for a nut at one end and headed at the other. Before the rod is passed through the holes in the timbers an iron washer is put on, and before the nut is put on a similar washer is placed on the threaded end. The washers will provide a larger bearing surface for the rods and prevent their cutting into the timbers. Where such ore bins are placed against side hills it may be necessary to vary their construction to conform to the slope of the hill, but in all cases it is well to give the floor a slope, in order to assist the loading when shipments are to take place.

TRANSPORTATION OF ORE

SURFACE TRAMWAYS

70. General Considerations.—At the smaller gold and silver mines, the transportation of ore and supplies is usually accomplished either with wagons or with pack animals. In the case of gold and silver mines having their own mill, a car track should connect the mill and the mine wherever this is possible. In some cases, ore and supplies are necessarily transported in wagons. This is true in mountainous countries, for a gold or silver mine may have a large output without having to transport much weight for long distances, the ore being first reduced to small loads of valuable metal. If transportation by means of wagons proves too expensive, or if, owing to the character of the country, a railroad is out of the question, either an inclined plane or an aerial, wire-rope tramway may be employed. In most large metal mines direct railroad connections are made with the mines in order that supplies may be delivered on cars and the ore taken away in the same manner.

71. Gauge of Tracks.—Where the mine has tracks of its own on the surface it is well to adopt a uniform gauge.

This is especially true where the mine cars are brought to the surface. For small railroads, the gauge of the track is sometimes measured as shown in Fig. 62, which illustrates the standard gauge for industrial railways; as will be seen, the flanges of the wheels are outside, the rails being between the flanges. The gauge of most mining railroads

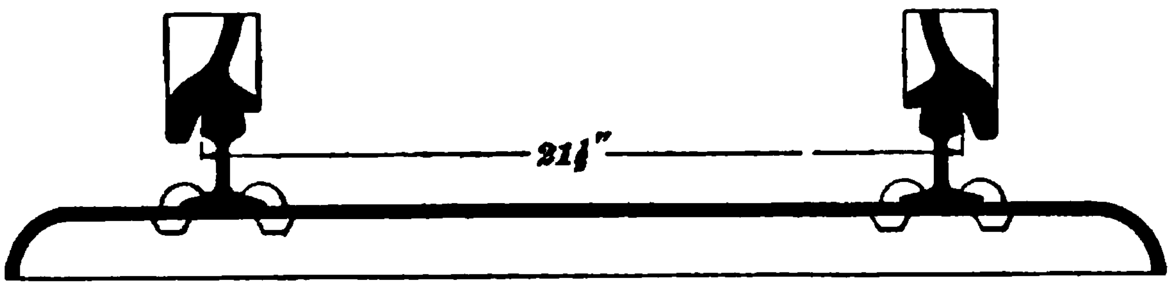


FIG. 62

is measured as shown in Fig. 63. The standard railroad gauge in the United States is 4 feet 8½ inches; it is important to remember this, since railroads seldom do the grading for side tracks leading to mines. At some mines the same gauge is employed for the skip road and for the surface lines, the tracks in the underground drifts being of a narrower gauge. The usual track gauge for vein ore mines of medium width is 18 inches, with the wheel flanges inside the rails. In large deposits, such as the iron-ore mines of the Eastern

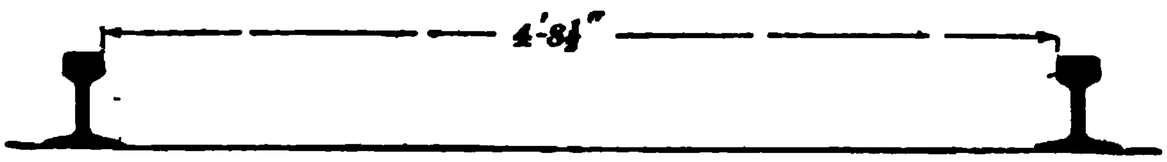


FIG. 63

United States, the usual track gauge is 36 inches. If side-dump cars are used in the mines, the track gauge must be narrower than where end-dump cars are used. In the former case, the car body must be raised higher as the gauge is increased, in order to clear the wheels and obtain an angle sufficient to discharge the ore. It is of course understood that the higher the car bodies are raised, the more top-heavy the cars become, and the more difficult they are to handle.

72. Uniform Track Gauges at Mines.—It is well to keep all cars and tracks of one pattern, since switches designed for cars having wheels flanged outside are not suitable for cars with wheels flanged inside. If all the cars

have wheels flanged inside, the narrow-gauge cars may be run over the same track as the wide gauge by laying a third rail, so that the narrow-gauge cars will follow one of the old rails and the extra rail.

Where cars are brought from the mine in cages, or where the same style of cars is used in the mine and at the surface, the track gauges should be the same in order to make the cars serviceable in both places. Skip tracks should always have a gauge at least 32 inches wide, as a skip's stability on a slope increases with the width of the skip and its decrease in height.

73. Mine Rails.—Iron or steel rails are used for mine tracks, although in some mines it is thought to be economical to use wooden rails on account of the rapid corrosion of metal rails by acid water. Iron or steel rails are spoken of as weighing a certain number of pounds to the yard; the rails used at ore mines vary in weight from 12 to 40 pounds per yard. Cars that weigh, loaded, as much as 2,000 pounds should run on rails not lighter than 18 pounds per yard. Frequently, cars of this and greater weights run over 12-pound rails; but such tracks soon become very uneven and irregular, and the cars frequently leave the tracks. It is not economical to use rails so light that they will spread and bend. Outside of mines the cars are frequently moved at higher speeds than inside, and hence rails from 20 to 40 pounds should be employed, according to the speed and weight of the cars. The lawful rate for hauling cars underground in Pennsylvania is 6 miles per hour, and this speed should never be exceeded in any mine; on the surface there is no lawful speed regulation. Light rails should have ties placed closer than heavier rails. The distance of ties between centers varies from 18 to 36 inches, and while the latter is given as the maximum for mine roads, it is too great a distance, even with wide-faced ties.

74. Car Wheels.—The larger the diameter of the car wheel used, the easier the car will run. The average mine-car wheel is 12 inches in diameter, but at times it is made

as much as 18 inches in diameter. Mine-car wheels are usually made to turn on their axles, but frequently they are pressed on the axles, in which case the cars are supplied with journal-boxes. The journal-boxes are sometimes outside the wheels, as in railroad cars, and sometimes inside the wheels. The hubs of loose car wheels wear fast, while with wheels fixed on the axle, the journals wear. The journal-boxes may be rebabbited, but the wheels when worn must be discarded. In order to obtain a short wheel base, for the purpose of taking sharp curves, while retaining a carrying capacity of 16 to 22 cubic feet, the ore cars of the kind shown in Fig. 59 are used. Where a cage or end-dumping apparatus is used, the truck may be wider, the wheels of larger diameter, and the height of the car above the rails less for a given capacity.

It is usual at ore mines, when newly purchased cars are received, to cover their bottoms with 1½-inch or 2-inch planks, as a measure of economy, since hard rock wears the steel away rapidly.

75. Grade for Inside Tracks.—The grade rule for tracks in mine levels is to give them sufficient rise to allow water to run away, say 1 in 200. It is usual to have the grades arranged to favor the loaded cars. Where it is intended that the cars shall run by gravity, away from the loading chute, for instance, the necessary grade will depend somewhat on the construction of the cars, since some mine cars run much harder than others, as explained in *Mine Haulage*. For loaded cars, a grade of from .75 foot to 1.25 feet in 100 feet will usually be sufficient, while for empty cars it may require a grade of from 1 foot in 100 feet to 2.25 feet in 100 feet to make the car start and run by gravity. The design of mine cars should be such that the car runner can control his loaded car if it runs by gravity, and push the empty car back easily at the ordinary speed of walking. These conditions apply equally to a train of several cars, except that friction brakes should then be supplied to the cars.

76. Roadbed.—Where a steam, electric, or compressed-air locomotive is to be used, the track should be laid with good ties, but the embankments and ballasting do not require the care that is usually bestowed on large commercial railroad lines, where the speeds are very much greater. The spaces between rails should be filled sufficiently to keep them free from water, and the ties should be well tamped under the ends. The ease with which cars can be moved over good roadbeds more than compensates for the extra work in making them.

77. Grade for Outside Tracks.—On outside tracks, if the cars are run by gravity, the same inclination, say 1.25 per cent., will answer, provided the tracks are kept clean. This is gradually reduced toward the dumping point in order that the cars may be easily stopped and accidents prevented. Broad-gauge tracks leading to ore chutes should never have less than 1.25-per-cent. grade, and even then the tracks must be kept clean to lessen the amount of prying required to make the cars start.

78. Switches.—Fig. 64 is a diagram of a turnout or switch. This particular switch is known as a **stub switch**, because the movable rails *a*, *b* are cut off square at the ends. As the movable rails *a*, *b* are placed, cars going to the right

would pass on to the rails *d*, *c*; but if the rails *a*, *b* were thrown into a position to break connections with the rails *d*, *c*, the cars would pass on to the curved rails *e*, *f*. The movable rails are thrown by a lever, which may be on the ground or upright on a stand. The points *a* and *b* shown in Fig. 65 form a switch

that furnishes an unbroken track. This kind of switch is known as a **split switch**, and is the one usually adopted on main track lines. The shifting rails are held at one end by being bolted to the rails *a'* and *b'*; however, they are made long enough to be readily moved by the bridles *c*. In order that they may be shifted easily, the points rest on sheet-iron plates *d*.

79. Frogs.—Fig. 66 shows a rail frog, not much used inside mines except where there is mechanical haulage.

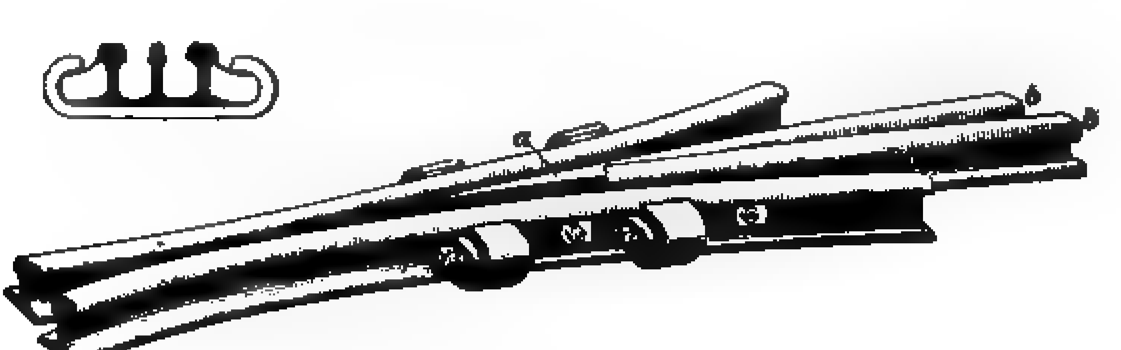


Fig. 66

Fig. 67 shows a spring-rail frog, which offers an unbroken track to trains passing over the rail *a b*, but not when passing

over the rail *c d* to go on to a side track; if, however, a car were traveling from *d* to *c* it would not be derailed. The rail *e* is movable, the car flanges pressing it outwards, but as soon as it is released it returns to its original position. In mines, rail frog points such as shown at *a*, Fig. 66, are used. These are usually made at the blacksmith shop, because mine

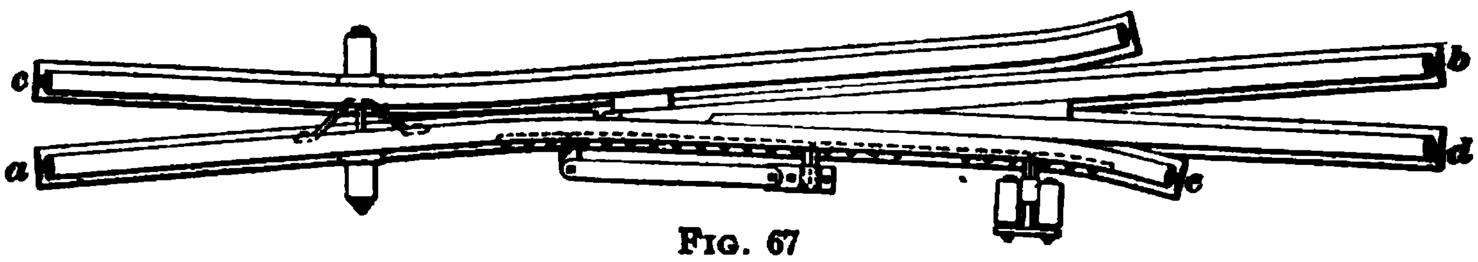


FIG. 67

curves are of small radius and the tongues *b* can be curved to meet requirements. Often, frogs are made of cast iron for both inside and outside work, but these are frequently disappointing, because the switches must be made to line with them, and in cramped spaces this is not always convenient; on the other hand, rail frog points can be made to accommodate the switches.

80. Turnouts, or **partings,** may be used for the passing of two trains, for side tracks, or for cross-levels; in any case, the object is to run cars from one track to another. The most common turnout is that shown in Fig. 68. The switch points are wedge-shaped bars of iron, Fig. 69, of the same height as the rails. The points are pivoted to the ties by a spike passing through the hole *a*, Fig. 69, and are connected to each other by a cross-bar *a*, Fig. 68, and bolts passing through the hole *b*, Fig. 69. This switch is readily moved from one

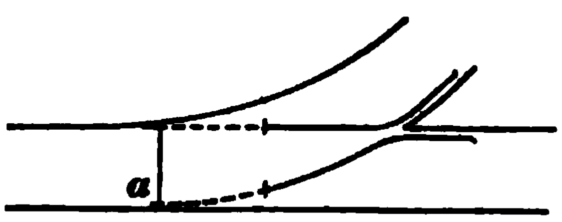


FIG. 68

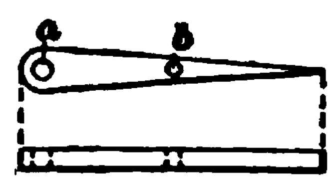


FIG. 69

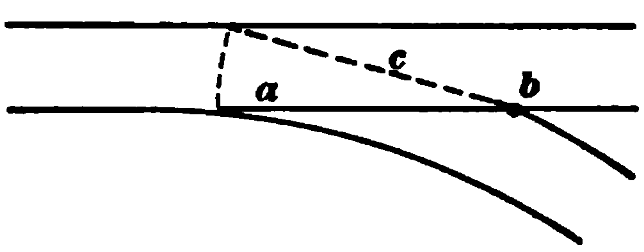


FIG. 70

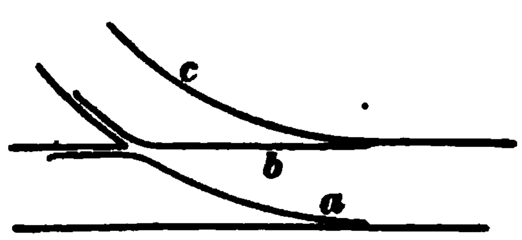


FIG. 71

rail to the other by the foot. Fig. 70 shows a parting having a rail *a* pivoted at *b*, so that it may be moved to the position

shown by the dotted line *c*. Fig. 71 shows a parting much used. The points *a*, *b* are spiked to ties in the position shown. When such a parting is reached, the car going to the turnout is pushed to one side, so that the front wheels shift to the rail *a*, and this crowds the wheel on the other side of the car over on to track *c*. The wheels following take the same rails.

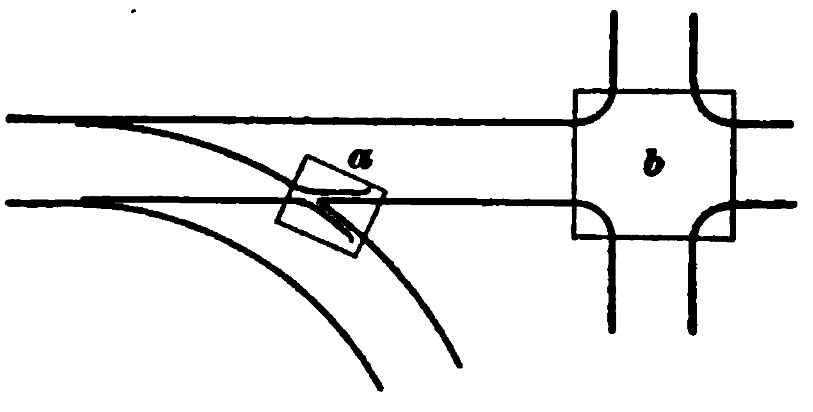


Fig. 72

Fig. 72 represents a somewhat similar switch, with a rail or cast-iron frog on a plate *a*. In mines it is often necessary to make turns at right angles to the main track, for which reason turnplate *b* is used.

81. Turntables are much used about mines, mills, and smelters. Fig. 73 shows one having ball bearings. These turntables are great improvements over turnplates, or turntables that are pivoted in the center. The balls, however, will wear the plates rapidly, for which reason considerable allowance is made at the bearings.

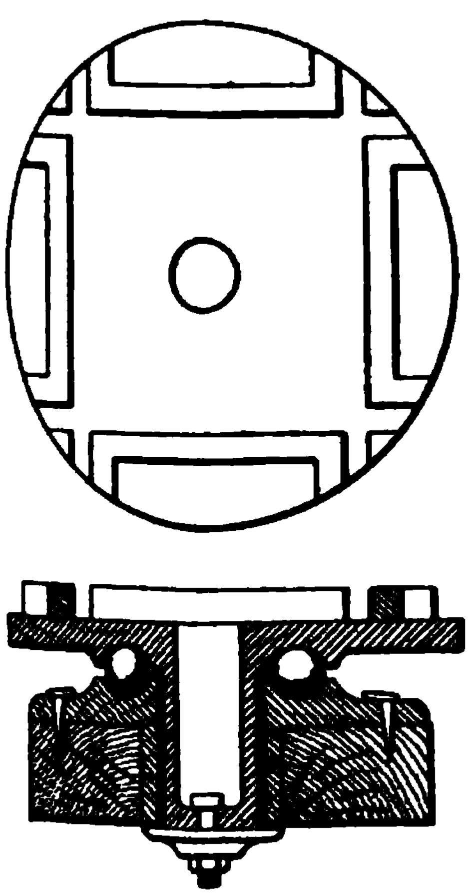


Fig. 73

82. Relation Between Ore Mined and Transportation.—The general arrangement of surface plants and the method of transportation adopted will depend to a large extent on the nature and extent of the deposit being worked. In the case of large iron, copper, gold, or silver mines it will be necessary to lay down quite an extensive plant and make preparations for handling large quantities of ore; while with small, irregular deposits, as at some zinc and lead mines, such extensive surface arrangements are out of place, and everything about the location must be of a more or less

portable nature. At some mines very little more than food and clothing has to be transported to the location, the timber necessary in the mining operations being obtained from the adjoining country. After the mill has once been erected and put in running order, it is to a large extent independent of outside communication, there being little to transport either to or from outside places. In such locations, wagon or pack-train transportation will answer; but where large amounts of concentrates or heavy ore are to be transported, a railroad, inclined plane, or wire-rope tramway is necessary.

ANIMAL TRANSPORTATION

83. Wagon Roads.—The engineer is frequently called on to select a mountain road up to the mine. A good method of procedure in such cases is to find the approximate height of the mine by the aneroid barometer, and pick out the most suitable ground during a walking survey, avoiding rocky tracks whenever possible. While it is desirable to have a uniform grade, it is not always possible, but grades for wagon roads should not exceed 8°, and then only for a short distance.

84. Loads.—Assuming that a horse can draw 3,000 pounds on a level, he can draw the following loads on inclinations:

Grade	Load in Pounds	Grade	Load in Pounds	Grade	Load in Pounds
1 in 100	2,730	1 in 60	1,800	1 in 30	900
1 in 90	2,700	1 in 50	1,500	1 in 25	750
1 in 80	2,400	1 in 45	1,350	1 in 20	600
1 in 75	2,250	1 in 40	1,200	1 in 15	450
1 in 70	2,100	1 in 35	1,050	1 in 10	300

85. Pack Animals.—In some instances it will be impossible to make wagon roads to the mines, in which case it may be possible to pack machinery and supplies on animals

backs. Pack animals can be given a load of about 25 per cent. of their own weight. A burro load is from 75 to

Fig. 74

100 pounds; a mule load from 150 to 220 pounds; a horse can carry from 200 to 400 pounds.

Fig. 75

The cayuse, or mustang, is not a large horse, but is strong in comparison with his size; the burro, however, is the more sure-footed and patient animal, and consequently is preferred

for mountain trails, at least in the Western United States and in Mexico. Fig. 74 shows a burro train packed with ore, ready to start for the mill or shipping point. Wire ropes are sometimes taken up the mountains by placing part of the rope on one animal, part on another, and so on, care being taken not to overload the animals. Fig. 75 shows a pack train ready to start into the mountains with wire rope. The end section of the rope is coiled on the foremost animal, the next section on the second, and so on, the stretches of wire rope connecting the sections swinging between the animals. This is almost the only practicable way of transporting long wire ropes to difficult places in the mountains.

86. Wagon Trains.—Where wagons are used for transporting ore, supplies, or other materials, it is a common practice to use one lead wagon, hitch two or three trail wagons behind it, and then draw the entire load by means of a team of from twelve to sixteen horses or mules. The advantage of this method is that one driver can handle a very much greater weight of freight than would be possible if the teams were divided up among the wagons and a driver provided for each. Another advantage is that, when a difficult hill is encountered, the wagons can be separated and all the teams employed for taking the individual wagons up the hill.

Wagons for hauling freight should have wide tires, in order to make hauling easy and avoid cutting up the road, as narrow tires on a heavily loaded wagon will do. Drivers soon learn the good and bad places in roads and just how much their animals can pull over them without injury.

87. Traction Engines.—In some localities traction engines are employed for drawing freight wagons in trains of from two to twelve. This method of transportation has been found especially suited to the borax industry in the deserts of Western America, particularly in California. Automobiles also are coming into use for transportation purposes, and will be of great service in dry and barren localities like Death Valley, Nevada, and other parts of the Great American Desert.

WIRE-ROPE TRAMWAYS

88. The wire-rope or aerial tramway is independent of differences in elevation and can pass over a country having a very rough profile, as shown in Fig. 76, which illustrates one of these tramways used for transporting ore from a mine to the mill. The conditions to be established in aerial tramways are straight lines from the loading to the discharging points, long spans, and sufficient carrying capacity. The tramway in Fig. 76 has a length of 9,000 feet with a span of 1,173 feet over the town of Wardner, Idaho. Its capacity is 400 tons daily. The difference in level between the mine and the mill is 713 feet. This is sufficient to work the tramway by gravity and furnish some power for hoisting.

89. The Bleichert System.—The Bleichert system was introduced into this country by the Trenton Iron Company. The loads are suspended from carriages that run on stationary cables and are moved by an endless traction rope. The track cable is suspended between towers of wood or steel placed at desired intervals. The descent of the tramway may be made so rapid and regular that loaded buckets in descent will exert sufficient power to overcome the friction of the ropes and pull up the empty buckets as well. In case the fall is not sufficient to work the system by gravity, or in case the tramway has an up-hill inclination that largely offsets the down-hill inclination, it will be necessary to make use of some motive power to move the loaded and empty buckets. The Bleichert system is known as the double-rope system, in order to distinguish it from the Halliday, or single-rope, system.

90. Bleichert Track Cable.—Any good wire rope will answer as the track cable, but the locked coil rope shown in Fig. 77 is recommended, since the carriage wheels will wear less than when running on twisted-wire rope. On comparatively level lines, this rope sustains the weight of the carriage, or bucket, and the ore, besides the weight of the traction rope between supports. The ropes may be obtained in

Fig. 78

lengths of from 800 to 2,400 feet, but such lengths are very heavy and cannot be transported readily, so that in most instances couplings are imperative.

FIG. 77

Since the rope must offer a uniform surface to the trolley wheels, the coupling shown in Fig. 78 is employed. It is made in two parts, with an opening in each into which the rope is inserted. The opening is funnel-shaped inside; hence, the

FIG. 78

ends of the wires are spread apart and the intervening spaces filled with wedges and conical rings driven in tightly. The halves are joined by a plug with right- and left-hand threads.

91. Tension of Wire Cables.—When a wire rope is suspended between towers of an aerial tramway, it will sag from its own weight and form a catenary curve. At the center of the span there will be a horizontal distance where there is tension due to the weight of the rope on each side of the center. If the cable is loaded the tension will be increased. The deflection will increase with an increase in span, and hence the tension will increase correspondingly.

The tension at the point of greatest deflection, where the cable becomes horizontal, equals the load between the points at which a horizontal line from the top of the lower tower intersects the cable, multiplied by the distance between these points of intersection divided by eight times the deflection below this line. The tension at either tower equals the

square root of the sum of the squares of the horizontal tension so obtained, and of the load between the tower and the point of greatest deflection. When the towers are on the same level this becomes:

$$\text{Tension at center of span} = \frac{\text{total load} \times \text{span}}{\text{deflection of cable} \times 8}$$

$$\text{Tension at towers} = \sqrt{(\text{tension at center})^2 + \left(\frac{\text{total load}}{2}\right)^2}$$

$$\text{Length of catenary span} = \text{span} + \frac{(\text{deflection of cable})^2 \times 8}{\text{span} \times 3}$$

92. Deflection of Wire Cables.—The following general formula will be useful in determining the deflections corresponding to a given tension at all points of a suspended cable, either sustaining a concentrated load or without load, and also the tension corresponding to given deflections.

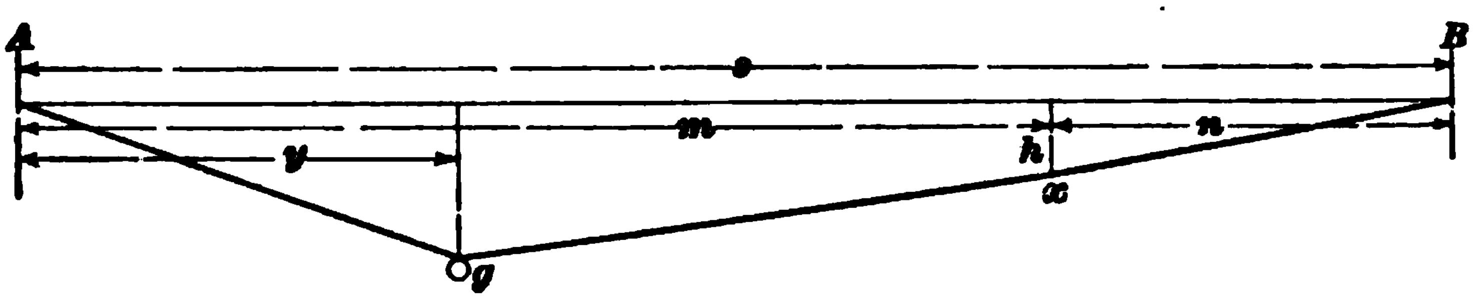


FIG. 79

In Fig. 79, let s = span, or distance AB, between supports;

m and n = arms into which span is divided by a vertical through point of deflection x, m representing the arm corresponding to the loaded side;

y = horizontal distance from load to point of support corresponding with m;

w = weight of the rope per foot;

g = load;

t = tension;

h = required deflection at any point x.

All measures are to be taken in feet and pounds.

Then; for deflection due to rope alone,

$$h = \frac{m\,v\,w}{2t} \text{ at } x, \text{ or } \frac{w\,s^2}{8t} \text{ at center of span.}$$

For deflection due to load alone,

$$h = \frac{gny}{ts} \text{ at } x, \text{ or } \frac{gy}{2t} \text{ at center of span.}$$

If $y = \frac{s}{2}$, then $h = \frac{gn}{2t}$ at x, or $\frac{gs}{4t}$ at center of span.

If $y = m$, then $h = \frac{gmn}{ts}$ at x, or $\frac{gs}{4t}$ at center of span.

For total deflection,

$$h = \frac{wmns + 2gny}{2ts} \text{ at } x, \text{ or } \frac{ws^2 + 4gy}{8t} \text{ at center of span.}$$

If $y = \frac{s}{2}$, then

$$h = \frac{wmn + gn}{2t} \text{ at } x, \text{ or } \frac{ws^2 + 2gs}{8t} \text{ at center of span.}$$

If $y = m$, then

$$h = \frac{wmns + 2gnm}{2ts} \text{ at } x, \text{ or } \frac{ws^2 + 2gs}{8t} \text{ at center of span.}$$

NOTE.—If the tension is required for a given deflection, transpose t and h in the formula just given.

93. Limits of Span.—The cables of wire-rope tramways must be drawn tight in part by mechanical means; but in using such tightening devices the rope must not be overstrained in order to reach the desired deflection. Wire-rope tramways have their loads distributed over the entire line, and a load on one span balances a load on another. The limits of the span depend on the contour of the ground, and consequently must be longer over ravines than on levels; but it is not advisable to erect high towers in order to obtain long spans, for the reason that they are expensive, are more liable to damage, and cause more tension on the rope.

94. The **traction rope** may be ordinary wire rope; on comparatively level lines it is subjected to very slight strains. On slopes, however, the weight of the load produces a tension that increases with the degree of inclination, just as in the case of hoisting ropes.

95. The Supports.—The towers for the cable and the traction rope may be of wood or iron. Their height and

strength depends much on the contour of the ground. On level ground their height will depend on their distances apart, which may be from 150 to 200 feet. Fig. 80 shows a steel tower for level ground; *a* is an arm for the cable from which the bucket is suspended; *b*, *b* are the rollers that carry the sag in the traction rope; *c*, *c* are the guides to prevent the traction rope from getting under the traction-rope arms. Where a tower is above a ridge and the inclination is sharp, rail stations are introduced; these consist of a series of bents supporting rails that overlie the track cable and save it from excessive wear. Wooden towers are not so expensive as steel and often answer every purpose.

Fig. 80

Fig. 81 shows a side view and an end view of a wooden tower, *a* being the carrier arm, and *b* the traction-rope arm. The size of the timbers for wooden towers varies with the design of the tower principally, but depends also on the rope tension due to the weight of the rope and its load.

96. The Bucket.—While wire-rope tramways are able to carry logs, rails, and other mine supplies, their particular

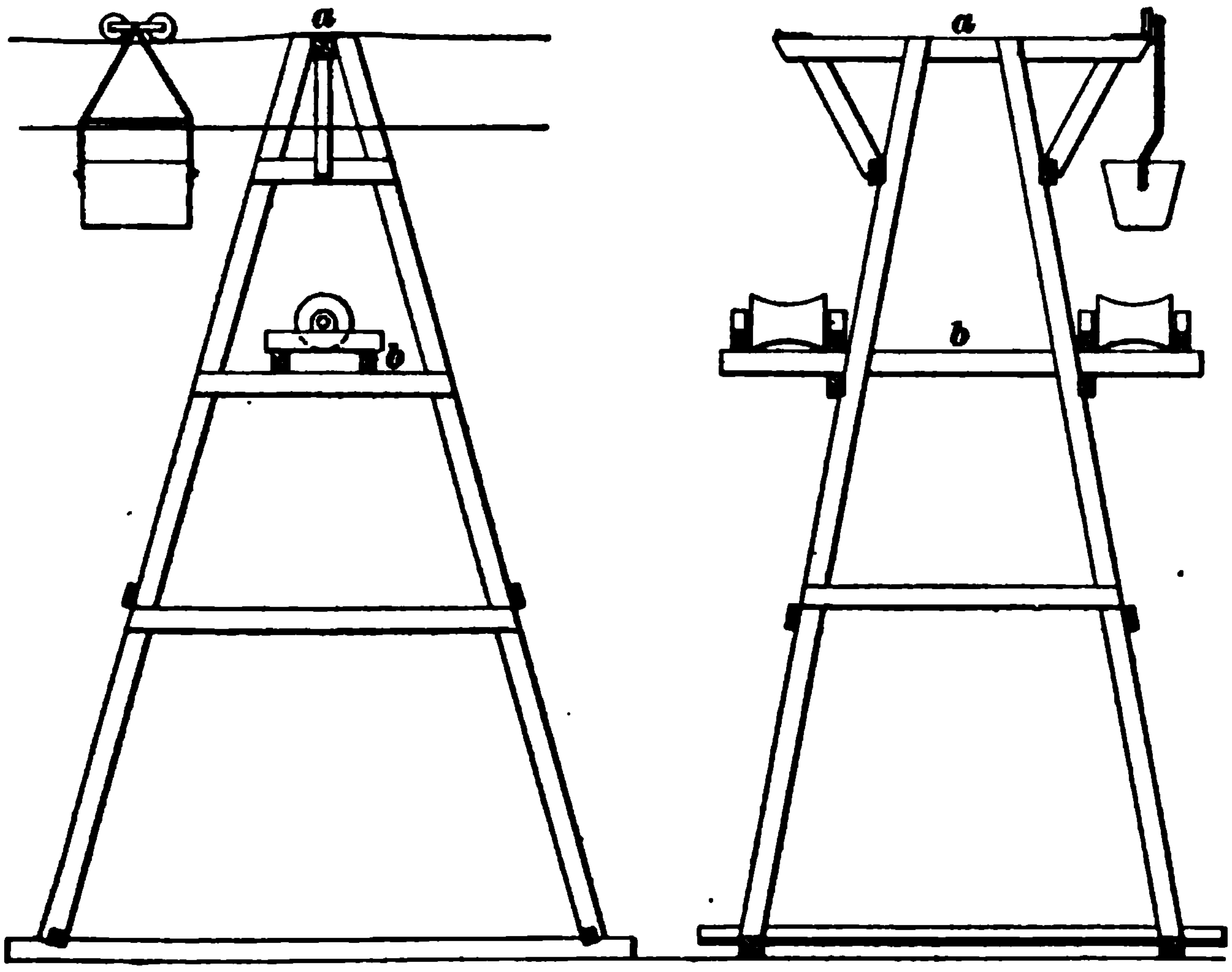


FIG. 81

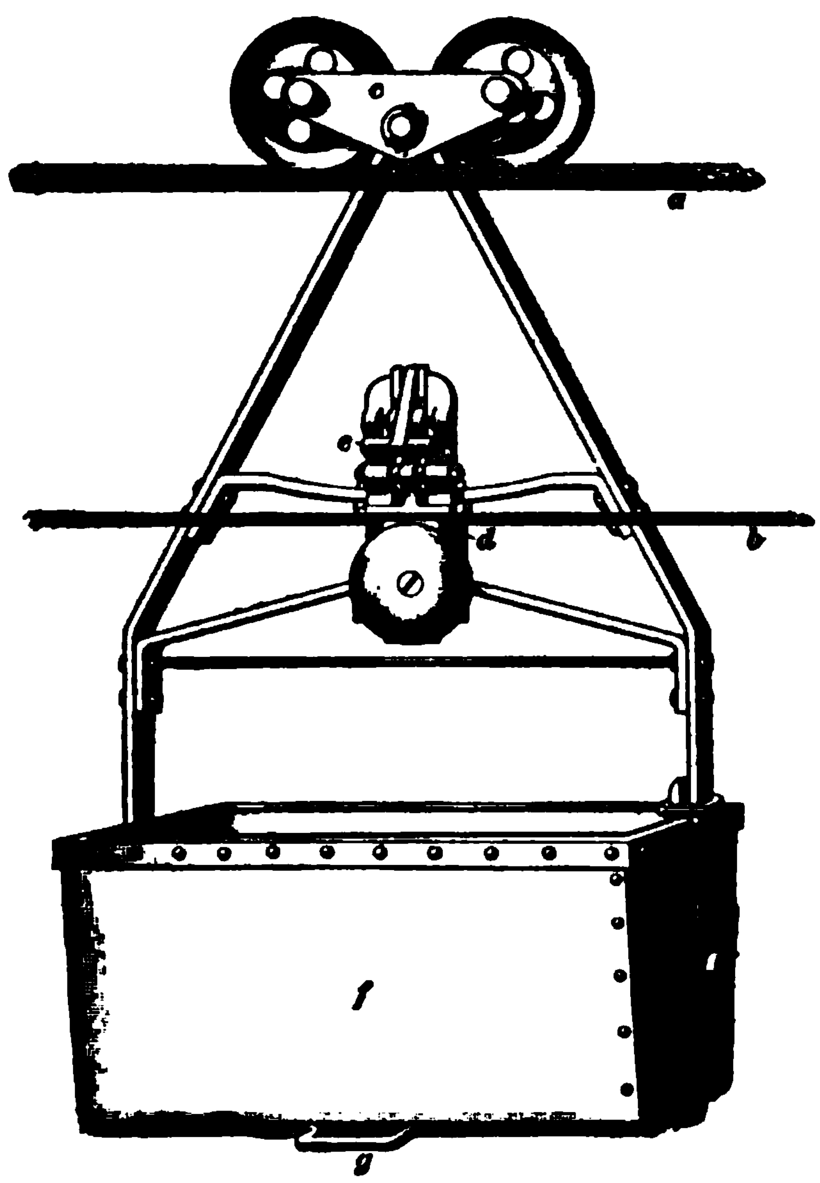


FIG. 82

object is the transportation of mineral. The buckets for this purpose are usually of steel, arranged so that they can be dumped automatically or by hand. In Fig. 82, *a* is the carrying cable; *b*, the traction cable; *c*, the trolley; *d*, the traction-rope grip, so arranged that it can be fastened or unfastened by lowering or raising the lever *e*; and *f*, the bucket, pivoted on bails at each end and supplied with a handle *g* to upset it.

97. Halliday Wire-Rope Way.—The Halliday wire-rope way consists of an endless moving wire rope that carries the ore buckets. The power for the railway is obtained from the loads when the inclination is 8° or more, and from external power when the delivery point is higher than the loading point, or when the line has less than 8° of inclination. When the inclination is considerable, sufficient power is furnished by the descending loads to carry supplies back to the mine as well as the empty buckets.

98. The **towers** for this system are not so expensive as for a double-rope system. The tower in Fig. 83 is braced for the top of a steep incline, the sheave wheels being four on a side, in order to prevent too great bending, and hence wearing, of the traveling rope at this point. Other towers are provided with two sheaves, one for the going and one for the coming buckets. The sheaves are provided with deep flanges to prevent the rope leaving the grooves.

Fig. 83

99. **The grip** is shown in Fig. 84. It consists of a shank *a*, the strap *b*, a key *c*, and a bolt and nut *d*. The strap surrounds the rope, and by means of the bolt, nut, and key holds the clip in position. The strap when worn out can be replaced with a new one. The key is used for tightening up the strap when the rope wears smaller.

FIG. 84

100. **The buckets** may be side or bottom dumpers; that shown in Fig. 85 is an automatic bottom dumper. In this case the bail *a* is bolted to the sides of the bucket *b* so that there is no swing. As the bucket reaches the unloading point, the lever *c* strikes the pin *d*, which unlatches the door and permits the load to be discharged, the door being hinged. After the load is discharged, the weight *e* on the end of a lever shuts the door and the latch holds it closed.

FIG. 85

101. **The Mechanical Loader.**—Fig. 86 shows an end and side elevation of a mechanical bucket loader *a*. It is placed in front of an ore bin *b* and receives the ore from the chute *c*. The device consists of a pendulum *d*, swinging on trunnions *e* about 20 feet above the moving cable. The loading box *a*, which contains when loaded enough ore to fill one bucket, is attached to the lower end of the pendulum.

The hopper has two sides, a back, and a sloping bottom; the front of the hopper is open. While the hopper is being loaded it is held between a guide and a fixed door, which closes the opening. The hopper box is released by the clip on the moving cable to which the carrier is suspended and which as it moves along raises a latch. When the loading box is released, the ore carrier is immediately under the

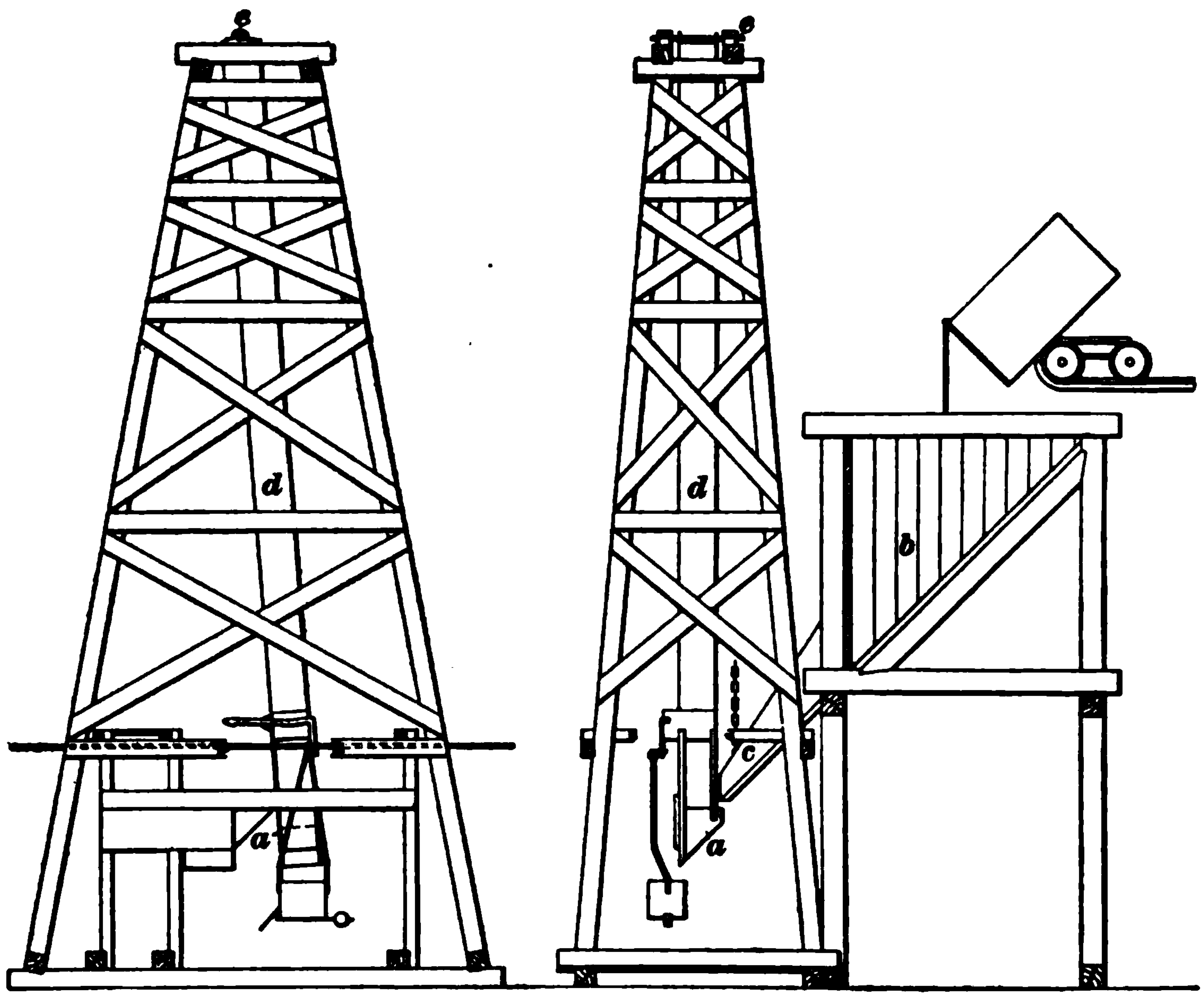


FIG. 86

nose of the loader box ready to catch the contents of the box. The clip on the moving cable then pushes the hopper out from behind the fixed door, at the same speed as the carrier, and thus opens up the front of the loader box and lets the contents pour into the carrier. The swing of the pendulum raises it sufficiently high after a few feet of travel to clear the rope clip, and the pendulum with the empty hopper swings back by gravity in between the guide and the bulkhead, ready to receive another load of ore from the ore bin.

102. Capacity of Wire-Rope Tramways.—The Bunker Hill and Sullivan wire-rope tramway averaged about 107 buckets, holding 732 pounds each per hour; for 10 hours this would be 390 tons. It also conveyed back to the mine 10 cords of wood per day, and could carry more. The operation of the road by gravity also develops some power, which is used for hoisting purposes. There are 127 buckets on the line placed 140 feet apart, and it is on the number of buckets that the capacity of such tramways depends. If there were 254 buckets placed 70 feet apart, the capacity would be doubled, the rate of speed remaining constant.

The longer the tramway, the greater will be the number of buckets required to transport a given quantity of ore in a given time.

103. Cost of Wire-Rope Tramways.—The cost of wire-rope tramways of the double-rope system is somewhat more than for the single-rope system, and either will cost about as much to install as a mine railroad for the same distance along good ground. But wire-rope tramways are useful for almost inaccessible places, and under such conditions are not to be compared in cost with any other system of transportation. When the plant is installed, the cost of handling material, loading, dumping, etc. is about 10 cents per ton, but this can be reduced at large plants to 8 cents per ton. This price supposes wages to be $3 per day, which is at present the average in Western America.

104. Cableways differ materially from wire-rope tramways, since they are used both for hoisting the ore from the mine and for transporting it to the place of shipment. Fig. 87 shows the open-cut workings at the Tilly Foster iron mine in New York State. In the figure, cableways with their carriages and cars are seen. The carriages run out on the ropes a certain distance, carrying with them the car body; as soon as the carriage stops, the car body begins to descend into the pit, where it is deposited on a car truck and pushed to the loading place. When loaded, the car is pushed under the carriage, hooked to the hoisting cable, and hoisted until

it reaches the carriage. The hoisting rope at this stage becomes the haulage rope and moves the carriage over the cableway to the dump.

Fig. 87

105. The Carriage.—Fig. 88 shows a carriage *a* used on a cableway *b*. The loads are raised by means of the fall rope *c*, which also acts as a haul rope and brings the cars from a higher to a lower level.

Towers are usually placed on the surface above the pits. The carriage *a*, Fig. 89, runs on the rope *b*. Attached to this

FIG. 88

FIG. 89.

carriage is a fall rope *c*, which is connected with the drum of a hoisting engine at one end and with the bucket *e*, Fig. 88, at the other. The fall rope passes half around the pulley *p*, then half around the block *d*, up to pulley *f*, and then down through the block to the car at *e*. The traveling rope *g* moves the carriage to the dump, holds it there, and permits the rope *c* to tilt the bucket. The bucket is next lowered to its normal position, moved back by the traveling rope, and lowered into the pit.

MILLS AT MINES

REDUCTION PLANTS

106. Gold Stamp Mills.—Occasionally, ore is delivered directly from the mine to the mill, and in such cases special pockets for loading cars are advisable. This arrangement is economical, and is to be recommended for small mines remote from ore-shipping facilities. The best locations for such mills are hillsides, on account of the convenience of receiving ore from higher levels and allowing it to descend from point to point as it is needed. The cross-section, Fig. 90, shows the car dumping the ore over a grizzly. The ore that passes over the grizzly goes to the crusher floor, thence through the crusher to the ore bin, where it mixes with the ore that fell through the grizzly. The ore next passes through a gate into an automatic ore feeder that supplies the stamps. From the stamp-mill mortar the ore passes as pulp over the amalgamating plates, and from there to the concentrating tables—an arrangement not always necessary with free-milling ores, but necessary for refractory ores.

Fig. 91 shows a sectional elevation of a double-stamp mill; the stamps are placed in double rows, in order to increase the number of stamps under one roof and lessen the lengths of single line shafting. Beginning at the top, the grizzlies are shown at *a*, the crushers at *b*, the ore bin at *c*, the ore feeders at *d*, the stamps at *e*, the plates at *f*, and the launder at *g*, the latter leading to the settling tank or pond. Where

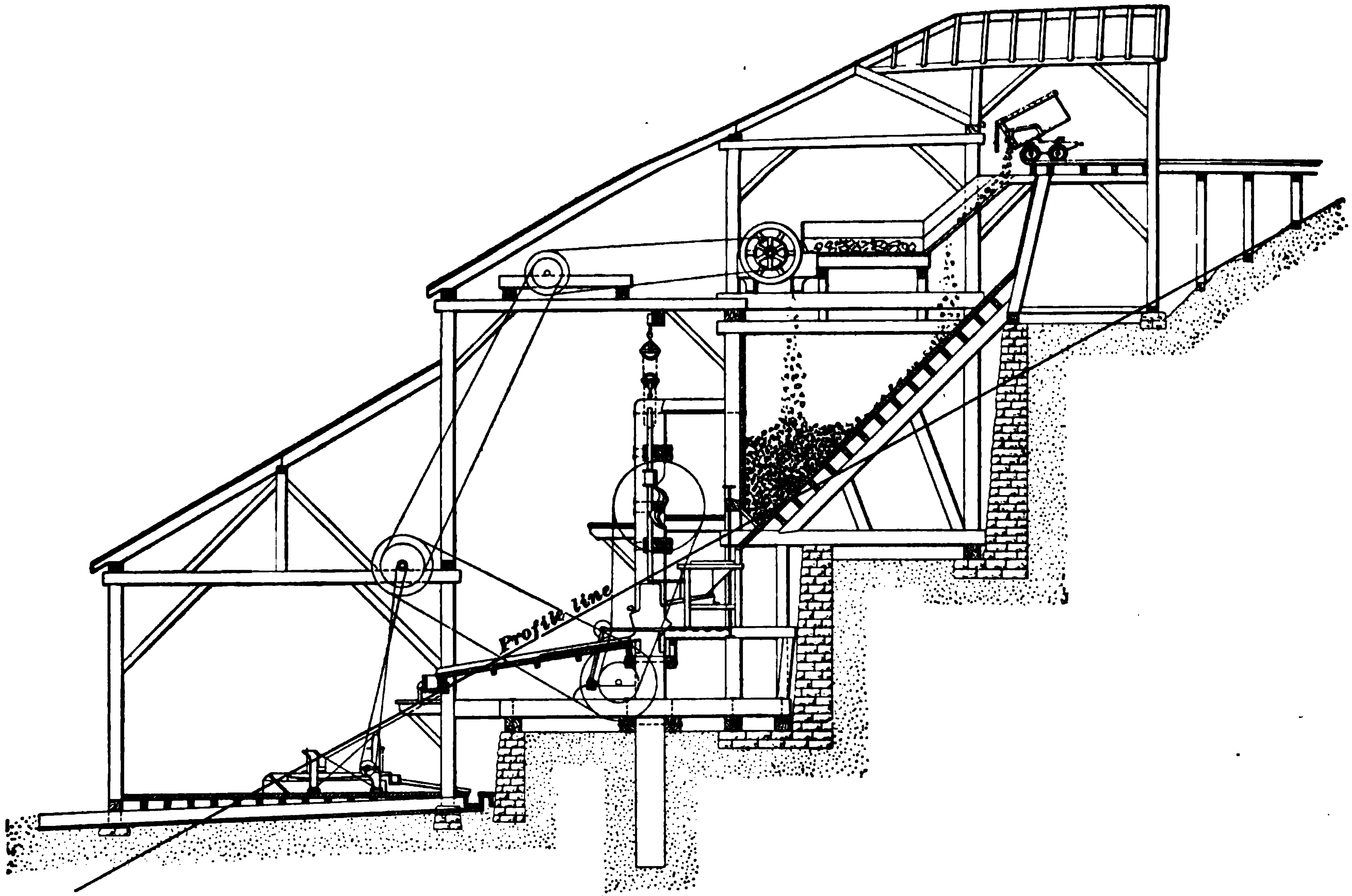


FIG. 90

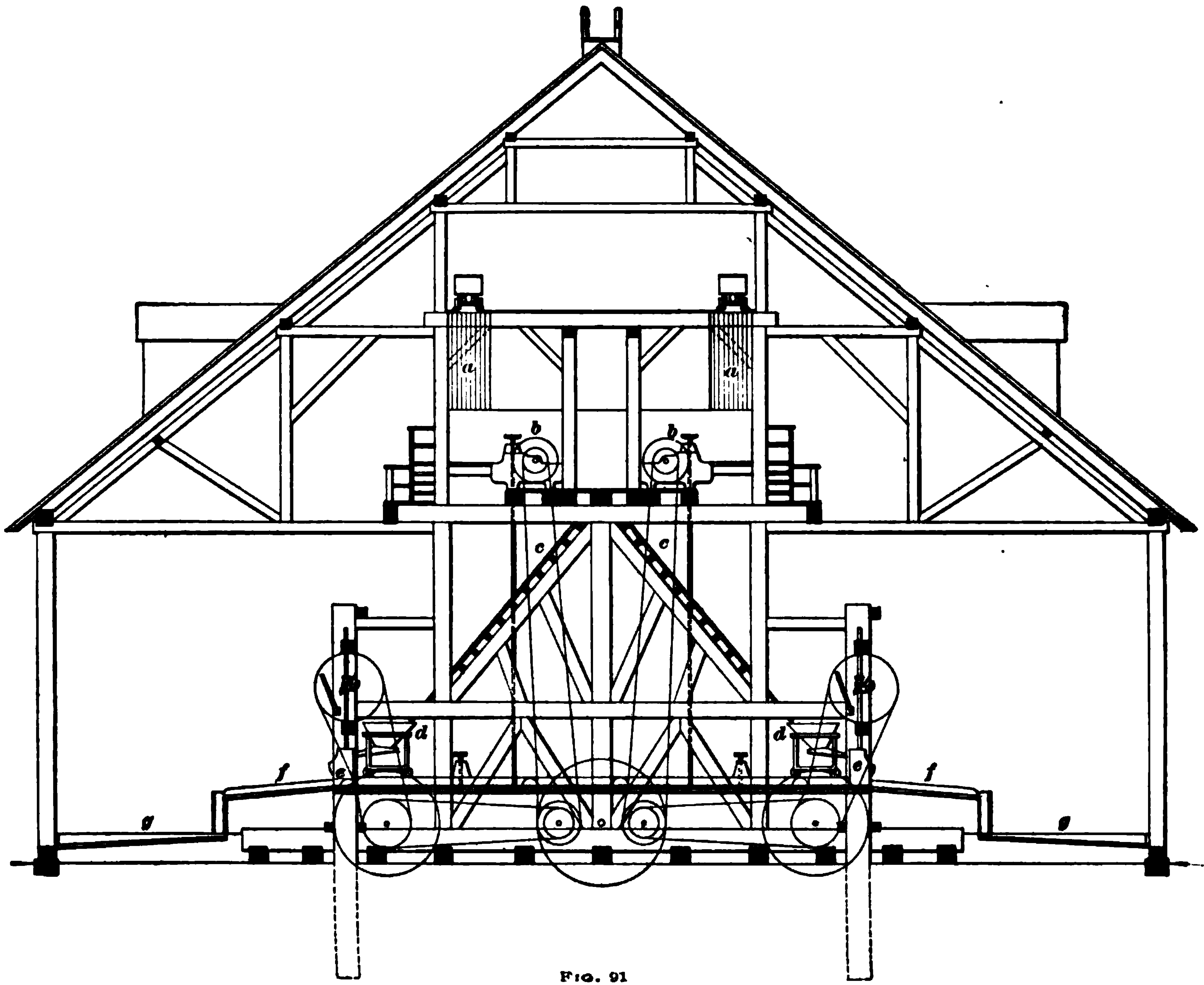


FIG. 91

a large number of stamps are to be used, and the ground is comparatively level, double-stamp mills are preferable to single-stamp mills.

Elevators and conveyers have proved so efficient at mills that gravity or side-hill construction is being abandoned wherever possible, particularly at large mills. Practically level ground is being picked out for mill sites. On such ground expensive retaining walls are avoided, as well as ore chutes, with their clogging and frequent repairing. Another advantage obtained by constructing mills on level ground is that employes are not compelled to tire themselves out by climbing and descending steep stairways; hence, they can accomplish a better day's work.

107. Silver Mills.—Silver stamp mills do not differ from gold stamp mills until after the pulp reaches the settling tanks. At this point the excess of water is drained off and the thick pulp shoveled in regular charges into amalgamating pans, in which it is worked for several hours in order to obtain the precious metals in the form of amalgam. The pan digestion being completed, the contents are run into large settlers, where quicksilver in the form of amalgam and free mercury settles to the bottom. The quicksilver and amalgam are at once separated from the rest of the pulp by a trap, and finally the amalgam is separated from impurities in the clean-up pan. A wet-crushing silver mill, such as has just been described, is shown in Fig. 92. In the figure, *a* is the grizzly, *b* the rock crusher, *c* the ore bin, *d* the ore gate, *e* the automatic ore feeder, *f* the stamp battery, *g* the battery plate, *h* the launder leading from the battery plate to the settling tank *l*, *m* the amalgamating pan, *n* the settling pan, and *o* the mercury and amalgam trap.

108. Dry-Crushing Silver Mill.—The dry-crushing silver mill is intended for refractory ores that must be roasted prior to amalgamation. It differs from the mill described by having the drier placed between the ore crusher and the stamps, which in this instance crush dry. The ore may be run in chutes lined with sheet iron from the driers

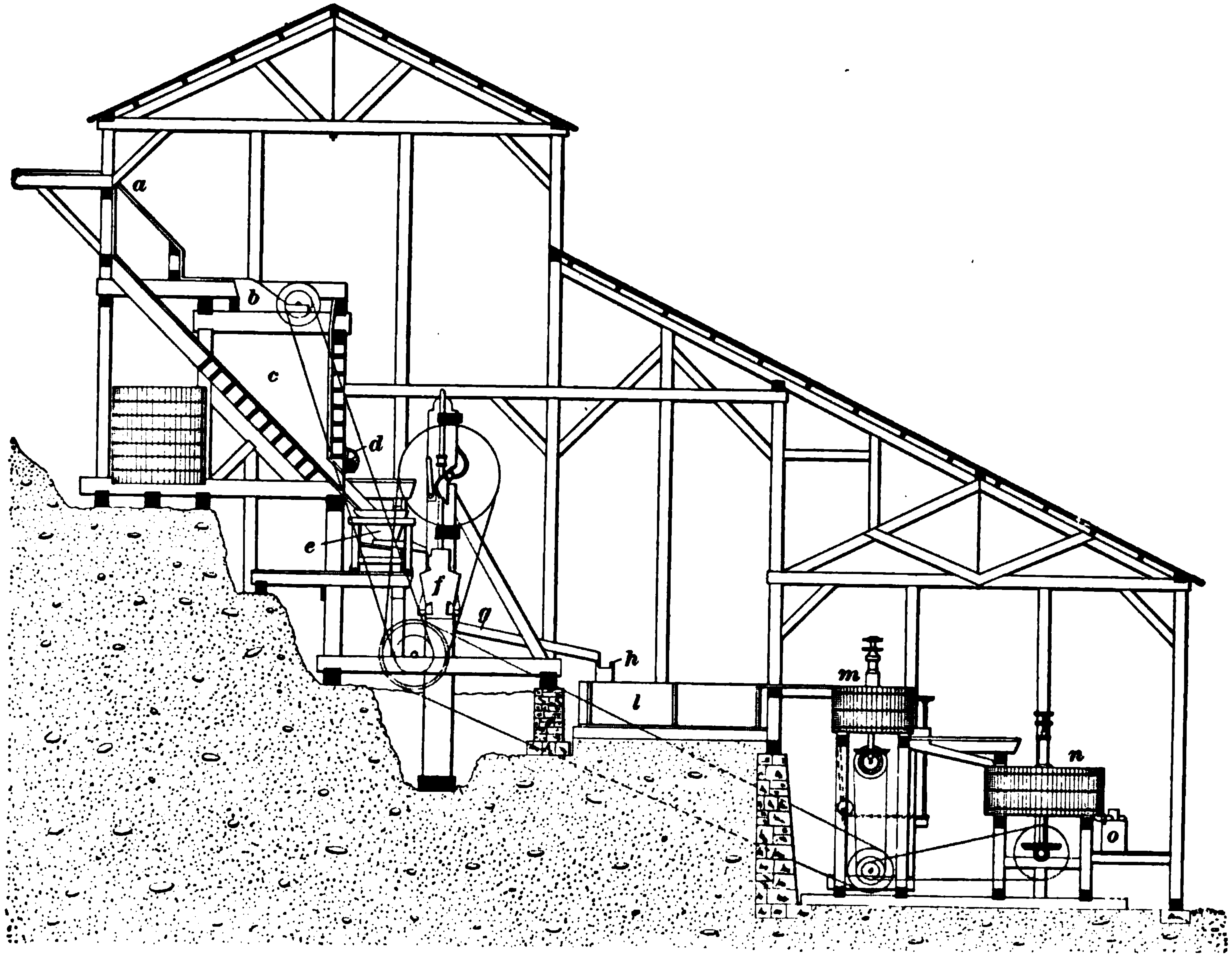


FIG. 92

FIG. 8

to the automatic feeder. The pulverized product from the stamps is carried by scraper or screw conveyers to bucket elevators, which discharge into the hopper of the roasting furnace. The ore is the furnace is desulphurized and also chloridized by the addition of salt, thus preparing the pulp for what is known as *barrel amalgamation*.

In Fig. 93 is shown a dry-crushing silver mill in which *a* are the grizzlies; *b* is the crusher; *c*, the ore bin; *d*, the drier; *f*, the stamp battery of twenty stamps; *g*, the screw conveyers, one on each side of the battery (the mortar in this case being of the double-discharge pattern); *h*, the furnace; *i*, the amalgamating pans; and *j*, the settlers.

At some dry-crushing mills, the ore is broken by rolls and properly sized for the furnace by means of rotary screens. One great objection to this method is the dust created by the several crushings and resizings; this dust can only be recovered by the use of exhaust fans, blowers, and settling chambers, and even with these precautions considerable loss will occur through valuable dust settling on belts and journal-boxes with destructive results. Notwithstanding these disadvantages, the system produces an increased amount of fine ore in a given time, though at the expense of power.

109. Elevators and Conveyers.—Since the introduction of elevating and conveying machinery, the handling of material at mines has been almost revolutionized. The cost of handling has been lessened, thereby increasing profits and making it possible to treat and handle large quantities of material. Conveying and hoisting machinery includes bucket conveyers, wire-rope tramways, elevated railways, cableways, and cantilever cranes. These numerous devices have modifications that permit them to be adapted to the work to be performed. The introduction of this class of machinery has made it possible to place mills on ground that would at one time have been considered unfit for the purpose. If the mills are large and the cost of power is not excessive, they may be situated on level ground and still the cost of handling

and treatment may not be more excessive than if situated on the side of a hill. Occasionally, mills are seen almost entirely hemmed in by waste rock and tailing piles, so that it seems as if they would be covered over if work were continued. Owing to the improved machinery for handling waste tailings and rock, such a state of affairs can now be entirely avoided. Conveying and hoisting machinery is used inside the mills as well as outside; and its flexibility permits it to be applied to the various new conditions that are continually arising.

Fig. 94

Fig. 94 shows a group of mills and head-frames in the Galena-Joplin, Missouri, district. The high towers appearing in the illustration are for elevating tailings from the mills and despatching them to the dumps, which also are shown.

POWER PLANTS

110. Steam Power. Besides the power required for hoisting ore, there will probably be needed power for air compressors, electric

lights, mill machinery, and pumping machinery. If the power required is steam, arrangements should be made for a central boiler house, from which steam can be transmitted to the various buildings by suitable pipes, covered to prevent loss of power by radiation and condensation. This central boiler plant would not be profitable if the buildings containing the engines were far apart. If possible, the boiler house should be so situated that fuel can be dumped into its full storage bins direct from the cars; and these bins should be large enough to contain a supply for at least 2 months. At some mines the boiler plants are quite elaborate, and if the plant is a large one it pays to make them so. Fig. 95 shows a mine boiler plant fitted with return-tubular boilers, mechanical stokers *a*, and coal hoppers *b* for feeding coal to the stokers. The boiler plant is about 200 horsepower and is attended by one man. The coal *c* descends to the floor by gravity from the coal bin outside, and is shoveled into the hoppers by the fireman. In larger plants, the hoppers are sometimes fed by conveyers, doing away entirely with handling coal between the bin and the furnaces. The ashes are removed by scraper lines in the bottom of the ash-pit, running at right angles to the front of the boiler.

111. Water-Power.—Some mines are so located that water-power is available for the generation of compressed air or electricity. It is usual now to employ an impulse wheel or turbine for power, since such waterwheels, where the height of fall available is considerable, are capable of affording great power.

112. Electric Power.—Electric power can be transmitted long distances from the place where it is generated, and can then, by the use of suitable motors, be converted into mechanical power for hoisting, drilling, and pumping. The installation of electric plants costs more than steam or compressed-air plants, but such plants are less expensive in the long run where water-power is available.

An electric-power plant also furnishes light, which is much needed about mines and mills; often it is considered advisable

to install electric plants for light alone, using steam to generate the electricity.

Electric mine drills have not yet proved satisfactory; but sometimes electric motors are connected to air compressors for the purpose of furnishing compressed air to rock drills.

113. Compressed-Air Power.—Compressed air may be transmitted to a considerable distance from the compressor, and in many instances is preferable to electricity. It may be used for running rock drills, engines, pumps, and driving electric motors. When the compressor is run by water-power, the cost is small compared with the cost of one run by steam.

Compressed-air transmission plants are now in use at a number of ore mines. Where a compressed-air transmission plant is employed at a mine, an electric-light plant may be maintained through a dynamo driven by an engine using compressed air. On account of the fact that an electric plant can furnish both power and light, and that compressed air is not generally required in mills or factories, electric-power transmission plants are very much more common than compressed-air transmission plants; but where the plant is installed for mining purposes only, the engineer will do well to figure the advantages of each system before deciding in favor of one of them.

114. Cornish Pumping Plant.—When pumping is done through two or more shafts, the pumps are sometimes operated by means of rods, as in the Cornish system of pumping. Fig. 96 illustrates an arrangement in which an engine is located between two shafts, *A* and *B*. The pumps in both shafts are operated by rods, which are connected to the ends of the bobs at *b*; the engine drives a large crank at *g*, and the rods operating both pumps are connected to this crank. The first rods *f, g* are pivoted at *f* and *g*; the connecting-rods that run to the shafts slide backwards and forwards over the rollers shown at *i*. At the other ends of the horizontal rods there are pivoted rods *e h*, which operate the bobs *c d e*. These bobs rest on piers at *b* and are

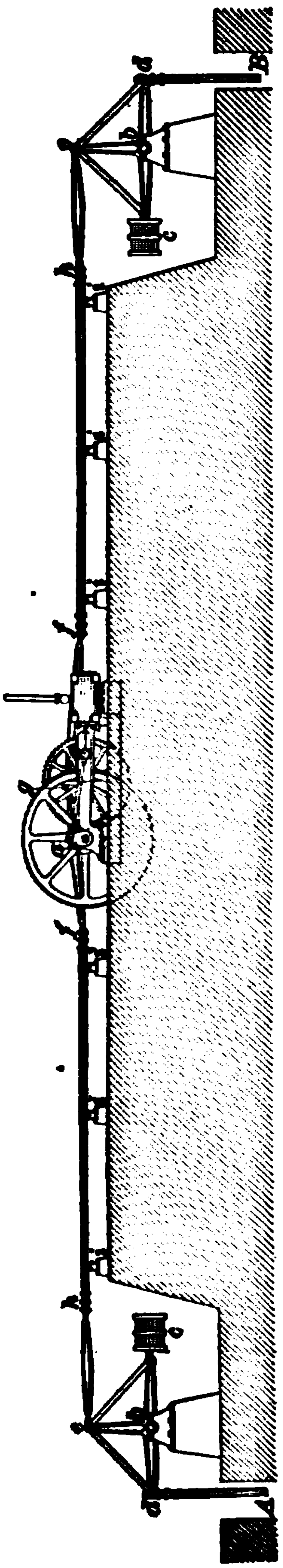


FIG. 98

provided with counterweights at *c*, which balance the rods hanging from the ends *d*. The result is that the engine is lifting one set of rods while the other is descending, and as the rods are balanced against each other the engine has only to lift the water and overcome the friction of the rods. Sometimes, the weights *c* are omitted and the rods connecting the points *e* and *g* are replaced by wire ropes. Wherever two shafts can be balanced against each other, this may be an economical arrangement. There are many instances where the Cornish pumping plant is used to operate the pumps in a single shaft.

115. Steam, Compressed-Air, or Electric Pumping Plants. Sometimes, electricity, steam, or compressed air is transmitted down the shaft to underground pumps. One advantage of this method is that there are no pump rods constantly moving up and down in the shaft. If either steam or electric pumps are employed, and the mine should become flooded with water, there is danger of not being able to start the pumps and so unwater the mine; while, if the Cornish pumps were properly greased and oiled before the mine became flooded, they can be started at any time. To a certain extent this is true of pumps operated by compressed air, especially when the exhaust of the latter is piped to the surface. The mining company may

also require a pumping plant to furnish water for general use at the boilers, at wash houses, and for protection against fire. Special pumping plants may also be required to furnish large volumes of water for the plant used in washing or concentrating the ore.

116. Shops.—Few ore mines have machine shops, it being more economical to keep duplicate parts of those machines most in use, such as pumps, compressed-air drills, compressor valves, etc. At some large mines, situated some distance away from public repair shops, machine shops on a small scale are a necessity. At those mines, however, which are only a short distance from public machine shops, a machine shop would be an expensive luxury. To do any large piece of work requires large machinery and tools; and to repair the stationary parts of machinery requires a foundry as well as a machine shop. A small machine shop, containing a drill press and lathe, a pipe cutter and threader, or a nut- and bolt-thread cutter, may be advantageously attached to the carpenter and blacksmith shop of any fair-sized mine; and while it is not a necessity it will be found to be a great convenience and time saver. The shop should be placed where power can be obtained from the regular power plant.

At mines having large stamp mills, the blacksmith shop should be supplied with a small trip hammer for the purpose of welding broken stamp stems and car axles, and for doing similar heavy work. Where a trip hammer is to be used, it is essential to have a good forge with wind blast in order to heat uniformly the parts to be welded. Since the carpenter shop, blacksmith shop, and machine shop together do not take up much space, they can be placed under one roof; the fact that they are more or less interdependent is an additional reason for keeping them together wherever possible.

117. Ore Dressing at Mines.—Power for concentrating mills may be taken from a central power plant; but, as a usual thing, unless the mill is driven by electricity, it has a power plant of its own. Picking-belt machinery requires considerable power, since the ore is crushed before it is

placed on the belt. Jigs require the ore to be crushed and sized before it is delivered to them. Concentrating tables necessitate finer crushing than jigs, and therefore more machinery.

In some cases, power is not required for ore dressing, the major part of the ore being of such a nature that only cobbing is needed. Picking tables may be moved by machinery or hand; if power is available for the purpose, it is generally used. Bucking boards or tables are now seldom seen at mines for ore dressing.

PREPARING ORE FOR MARKET

118. The plant for preparing the ore may be simply a crushing plant, which crushes the material after it is hoisted and before it passes to the storage bins or stock piles, from which it is to be loaded for shipment. The ore may be both crushed and washed to free it from clay, unless it carries gold and silver, in which case it becomes amenable to the ore treatment in practice by roasting to dehydration; or the plant may include sorting tables, belts, or floors, either associated with the crushing and washing plant or separate from them. The plant for treating the ore and recovering the metals is sometimes near the mine. This may be a complete concentration mill, a plant for the recovery of gold and silver, or, in exceptional cases, a smelter. When possible, these works are located near the head-house; but occasionally they are placed at some distance from the mine, where a better supply of water and fuel is available, or where there is a suitable dump for tailings or slag. Where the ore is merely to be crushed, the rock house may be a portion of the hoisting works; the skips are dumped over grizzlies, the fines passing into the pocket at once and the coarse material passing to the hopper of a crusher, which reduces the large pieces to a suitable size, the product of the crusher falling into the pocket that the fines passed into. If more extensive treatment is required, a more extensive plant becomes necessary. The general design of the works will depend on the

character of the ore. These matters come within the province of metallurgy, although some knowledge of them is necessary to the mining engineer, who may have to make plans for the arrangement of buildings to contain the metallurgical plant.

119. Railroad Connection.—In planning the arrangement of buildings at any mine, advantage is taken of gravity for the purpose of cheapening the handling of ore; and where the mine is connected with a railroad, as much of the plant as possible should be connected with the railroad also, so that material can be taken away from, or supplies delivered to, the most convenient place.

120. Compactness.—The plant should be compact without increasing the fire risk by having the buildings too near together. One advantage of such an arrangement is that the haulage of ore and supplies is reduced to the shortest possible distance, and the superintendent, foreman, or watchman can pass over the ground with ease. The company's offices, repair shops, etc. should be as centrally situated as possible, and if the mine owns the houses in which the men live and maintains a company store, the latter should be centrally placed, and the houses should be so arranged as to have good drainage, light, and ventilation. If possible, all buildings should be in reach of the company's water plant, so as to be reasonably safe against fire.

121. Handling Powder.—The powder house at a mine should be situated where it cannot be affected by blasts in the mine or by the jar of the stamp-mill machinery; its location also should be such that in case of an explosion the surrounding property will not be damaged. The magazine should be close to the railroad, so that the day's supply of powder can be brought from the magazine to the mine on a grampus. The powder should be moved in the daytime, when the men can see what they are doing and can signal cars out of the way. Powder should be handled at a time when there are no men traveling through the shaft or at the stations except those actually engaged in the work of

handling the powder. If the thawing is done underground, the powder can be taken to the underground magazine at once. Caps and fuse should never be kept in the same magazine with the powder; a special place should be provided for them. Where large quantities of powder are used, thawing rooms should be established at the mine and one day's supply of powder kept in them.

PRELIMINARY OPERATIONS

(PART 1)

INTRODUCTION

1. Scope of the Subject.—In this Section, it is assumed that the mineral deposit has been proved, by prospecting, to be of sufficient value to warrant the outlay necessary for a permanent opening. This may run in a horizontal, an inclined, or a vertical direction, and be excavated either in the deposit itself or in the adjacent country rock, depending on the inclination and character of the deposit and on the contour of the surface. Permanent openings should not be located without a careful consideration of the data obtained during the prospecting period, and of whatever information may be derived from an inspection of openings on adjacent properties. In case the prospect opening is made the permanent opening, its walls should be supported and every care exercised to make the opening permanent.

There is an objection to using data derived from the practice at adjacent properties; namely, that, as ore deposits are at best but irregular, it is not to be taken for granted, because the adjacent property has rich ore or good walls and easy mining, that the same conditions will necessarily prevail close by; for, as a matter of fact, changes frequently occur in a mine that require different arrangements to be adopted as mining is advanced. Various devices and methods will have to be adopted to support excavations, and these will differ according to the material penetrated;

so that in some cases the engineer will have little trouble while in others his ability to cope with difficulties will be put to a severe test. The scope of this subject, therefore, will include the various kinds of openings, as well as their location and the methods used in excavating them.

2. Adit.—A nearly horizontal passage from the surface by which a deposit is reached for exploitation, drainage, or transportation is termed an **adit**. An adit, therefore, is a *tunnel*, and if driven in the deposit is called a *drift*, but if driven to intersect the deposit at an angle is termed a *cross-cut tunnel*.

3. Drift.—The term **drift** has a double meaning. A drift differs from similar horizontal or nearly horizontal passages in that it is always driven in ore. A drift may be an adit when the latter is driven in ore, and it may be a nearly horizontal underground passage in the ore; it is from the latter meaning that the term **drifting**, in the sense of making passages in ore, is probably derived. This definition, however, distinguishes such a passage from a cross-cut, which intersects a deposit, or from a level, which may be either in or off the mineral. Frequently, the term drifting is applied to making nearly horizontal passages in rock, although such a definition is, strictly speaking, not correct from a mining standpoint. The coal miner calls an adit driven in a mineral a *drift*, and the adit mouth with him is the *drift mouth*.

4. Level.—An underground passage excavated along the course of the deposits, either in the mineral or in the country rock, is called a **level**. The term *passage* suggests that such excavations are used for transportation, which is, in fact, the case during some part of the life of the mine, and those cross-cuts that connect with the levels and through which mineral is transported to the shaft are termed *cross-cut levels*. A level may be driven in the hanging or in the foot-wall, as well as in the deposit, and in ore mines a series of levels are driven at regular intervals one above the other. This is shown in Fig. 1, where the ore is represented by the dark portion of the illustration. The white spaces at regular

intervals are levels driven in ore, while the passages that connect them with the shaft are cross-cut levels.

In order to distinguish one level from another, cross-cut levels are numbered 1, 2, 3, etc. from the surface down, or, if 100 feet apart, are called the 100-, 200-, 300-, etc. foot levels. Whenever a level is particularly rich or poor, or has any other noteworthy feature, the name of the man that did the driving is sometimes given to it. The distance apart of levels may be made too great, notwithstanding the fact that it is good practice to keep them as far apart as is consistent with economical and effective mining. The original plan in ore mines of placing levels 60 feet apart on the dip of the deposit has gradually been supplanted by a plan involving still greater distances. While this plan is apparently favorable, it is not entirely so, although based on the fact that the ore is blocked out into rectangles bounded above and below by levels, and along the strike by winzes or risers. This ore has to bear the expense of working and pay the cost of the cross-cut level to the shaft; consequently, the shorter the distance between levels, the greater will be the expense for dead work, as more cross-cuts will be required for a given depth. In Fig. 1, there are five cross-cuts, and if they are 60 feet apart, with the first cut 60 feet from the surface, they will reach a depth of 300 feet. If the cross-cuts had been spaced 100 feet apart, there would have been but three cuts necessary to reach the same depth and the money expended in dead work on the second and fourth cuts would have been saved.

FIG. 1

There is still another method of analyzing this subject. Suppose that the cost of driving levels and winzes is $8 per foot, and that the distance between levels is 60 feet and the length of the block is 100 feet; then, the cost of driving the winze would be $480, and of driving the level would be $800, or a total of $1,280. Continuing the calculation for the sake of comparison, assume that the vein is composed of quartz and is 3 feet wide; then, 1 ton will contain approximately 12 cubic feet of ore, or for the entire mass

$$\frac{3 \times 60 \times 100}{12} = 1{,}500 \text{ tons of ore}$$

To block this out would require an outlay of $1,280 ÷ 1,500 = 85⅓ cents per ton. Using the same figures, with the exception of the distance between levels—and making that 100 feet instead of 60 feet—the winze will cost $800 and the level $800, or a total of $1,600. The number of tons blocked out will be

$$\frac{3 \times 100 \times 100}{12} = 2{,}500$$

and $1,600 ÷ 2,500 = 64 cents per ton, or a saving of 21⅓ cents per ton.

In either case, it can be contended that the excavation may be in ore that returns profits and that therefore aside from the operation being a mere matter of saving money, it may contribute materially to the profits of the enterprise. For instance, in five blocks 60 ft. × 100 ft., there are 800 linear feet of excavation; while in three blocks 100 ft. × 100 ft. there are but 600 feet of excavation. Now, if there was a profit of $10 per foot from ore obtained in driving, then each 60-foot level would produce a return of $1,600; or the five levels would produce $8,000 against the $6,000 produced by the three levels, and the funds are more readily available. One must bear in mind, however, in such calculations as these that the ore in the block has to stand the expense of driving. Thus, in the 60-foot blocks, while the cost of driving, which the ore must bear, is 85⅓ cents a ton, the profits derived are equivalent to $1,600 ÷ 1,500 = $1.06⅔ per ton, or a net profit of 21⅓ cents per ton. The

cost of driving the 100-foot block is 64 cents per ton, which the ore must bear; but the profits derived from driving are equivalent to $2,000 ÷ 2,500 = 80 cents per ton, or a net profit of 16 cents per ton. Hence, it will be observed that it may be possible with 60-foot levels to work to better advantage than with 100-foot levels.

It is not altogether the cost of blocking out ore that favors the increased length between levels; in fact, the dead work entailed by driving the cross-cuts may be of such a nature as to eat up all the profits derived from mining with 60-foot levels. In the latter case, if the levels are driven 60 feet and the cross-cuts every 120 feet, advantages may possibly be obtained that will prove more beneficial than the 100-foot level plan. If the ore is pockety, or is in stringers that lie flat throughout the vein, there is not so much danger of missing the ore with levels that are moderately spaced. In deposits that dip less than 45° from the horizontal, mining may be expensive on long stopes, owing to the necessity that exists for a greater amount of handling and timbering.

There is still another matter for consideration; namely, a change in the dip of the bed between a pair of levels. The distance on the vein between levels is increased as the inclination is decreased, and is decreased as the inclination is increased; hence, any change in dip will produce a change in the cubic contents of the block of ore.

5. Cross-Cuts.—In ore mining, there are three adaptations of the term **cross-cut,** but in each case the excavation, no matter what its object, is driven at an angle to the course or strike of the deposit. When a tunnel is driven from the surface through barren rock, to intersect a vein, it is called a **cross-cut tunnel.** When nearly horizontal passages, such as those shown in Figs. 1 and 2, are driven from a shaft to the deposit, they are called **cross-cut levels.** When an excavation is driven from a level across a deposit for the purpose of prospecting, or into the walls of a deposit to look for parallel veins or ore bodies, it is termed a **cross-cut;**

in some instances, such excavations are very important. As an illustration of the importance of intelligent cross-cutting, it has been shown—particularly in the Keystone mine, Amador County, California—that the ore shifts laterally from one side of the deposit to the other, requiring cross-cuts to be driven several hundred feet. Some mineral deposits are in crushed areas of rock; hence, the mineral may suddenly disappear in one place and appear in another. In one case in Colorado, the mineral was suddenly lost where the wall rocks were smooth and regular. In hope of again finding the mineral, the vein was followed several hundred feet, when a cross-cut disclosed a valuable body of ore parallel to the vein proper. This is not an unusual case in Clear Creek and Gilpin Counties.

6. Cross-cuts are usually made in barren rock, if driven from a shaft to intersect the deposit. The smallest size that can be profitably driven is 6 ft. × 7 ft., and the average cost of such work is $8 per foot. If the rock is strong and tough, little timber will be required; on the other hand, if the rock is weak, timber may be needed the entire length, in which case the excavation should be made 7 ft. × 8 ft. in area. It will be noticed in Fig. 1, that the cross-cut levels increase in length with depth, and that each one must decrease the profits. For example, if the inclination is 30° and the levels are 100 feet apart, the first cross-cut will have a length of 173 feet, and the fifth cross-cut a length of 866 feet, or five times as much. This may be calculated as

Fig. 2

follows: sin 30 : sin 60 = 100 : x, or .5 : .866025 = 100 : 173. Again, .5 : .866025 = 500 : 866. If, therefore, the first cross-cut costs $800, the fifth cross-cut will cost $4,000. Accordingly, for deposits that have inclinations less than 60°, it is better to place the shaft in the position shown in Fig. 2, provided the rock conditions and surface contour will permit.

7. Shafts.—**Shafts** are excavations made downwards from the surface either in mineral or in country rock. In ore mining there are—without any good reason for it—two kinds of shafts; namely, those that run vertically from the surface and those that are inclined. Shafts are usually sunk in country rock, either in the foot-wall or in the hanging wall of the deposit, as shown in Figs. 1 and 2. Some veins are so nearly vertical that the shaft may follow them down. This arrangement is not to be recommended, however, if pillars of mineral must be left between levels for support, and if any disturbance caused by mining will throw the shaft out of line. Of the two positions, the foot-wall is preferable; but since the shaft will continue vertical while the deposit slopes away from it, judgment should be exercised in making a selection.

When a shaft is sunk in the hanging wall, it will at some depth intersect the deposit if that continues downwards. As the shaft approaches the place of intersection, the cross-cut levels become shorter, and after this point has been passed the cross-cut levels become longer. There is some advantage in this arrangement, particularly where ore decreases in value with depth; but there is danger of the shaft being moved out of plumb if the vein walls collapse after the mineral has been excavated.

8. Inclines.—**Inclines,** or **slopes,** are usually sunk on the ore deposit when conditions are adverse to sinking a shaft. Owing to the changes that occur in inclined deposits—such as rolling foot-wall or swelling hanging wall—it may be necessary to excavate part of these walls in order to obtain a uniform height for hoisting purposes. In some cases, where the inclination changes abruptly and then again

assumes its original direction, it may be necessary to lift up a considerable amount of foot-wall in order to obtain suitable tracks for the cars. Sometimes, it has been deemed advisable to excavate the incline entirely in the foot-wall, but recent improvements in slope skips have made it possible to follow the ore, even with irregularly dipping deposits, and without going to the expense of excavating rock; provided, of course, that there is a sufficient thickness between the walls for the cars. Whenever it becomes necessary to break walls on a slope, the foot-wall, except under extraordinary circumstances, should be given the preference. When a foot-wall is broken, the roof is not weakened, and any water that comes through the break follows the wall to the ditch below; on the other hand, when the roof is broken, water will probably flow through the break and drip to the foot-wall, thus making work at this point disagreeable and sometimes dangerous. Permanent slopes are not advisable for thick deposits, as subsidence due to mining operations will have some effect on them; furthermore, they must be carefully timbered. Shafts are better for permanent openings where the deposit is nearly vertical.

TUNNELS

DETERMINING FACTORS

9. Advantages of Adits.—It is not considered good practice to drive a long tunnel through barren rock in order to tap a deposit known to exist at the surface but not known to exist at the tunnel level. An adit drift can furnish most satisfactory arrangements for stoping ore to the rise, but requires the assistance of a slope or shaft in order to mine to the dip of a deposit. If there is not a large area of stoping ground above the adit, a shaft or slope is preferable, since the latter will be needed eventually; however, surface conditions may be such as to render an adit advisable even though a shaft is required—in which case the opening will be permanent.

When an adit is located some distance below the apex of the deposit, it may prove to be a very efficient opening, as any water that comes to it by gravity from above or is pumped to it from below will run away out of the adit mouth. The cost of hoisting under the same conditions will be decreased if the adit is used for transportation. These two factors may more than pay the cost of driving the adit, by the saving effected in fuel and labor, but before deciding on long adits the conditions should be carefully studied. At some mines, the cost of pumping is the largest item of expense; for instance, at one mine 17 tons of water is pumped for every ton of mineral mined. On the Comstock lode, water was encountered that was very hot and difficult to pump, for which reason the Sutro tunnel was constructed to tap the lode at a depth of 2,200 feet, an immense saving of pumping and hoisting being thereby effected. The water flowing from this tunnel contains sulphate of aluminum in such quantities that rocks are formed along its course.

Several tunnels have been driven for the purpose of unwatering mines, the Jeddo tunnel, in Luzerne County, Pennsylvania, being the longest single tunnel of this character that is on record. This tunnel is strictly a drainage tunnel, no mineral whatever being taken through it. The Newhouse tunnel, in Colorado, is another long tunnel, whose object is to drain mines, transport ore, and do prospecting work.

10. Size of Adits.—Since an adit is a nearly horizontal passage for transportation and drainage, an explanation covering the practice followed will also apply to similar excavations, no matter what their names may be. The size of adits depends on their object, and to some extent the character of rock in which they are driven. Single-track adits are given an area of 6 ft. × 7 ft. where no timber is required; 6 ft. × 8 ft. if the roof needs support; 8 ft. × 8 ft. if three-stick timbering is needed; and 8 ft. × 9 ft. if full timber sets are required. The differences in height and width are made necessary by the timbers and lagging, the unit area of 6 ft. × 7 ft. being considered the minimum for any adit.

Double-track adits are from 10 to 16 feet wide; in most ore mines, probably 12 feet would be the average, as this width affords room for the men to pass between the cars and between the cars and the walls. The heights of such excavations are governed by the size and thickness of the timbers required for the roof and the floor.

In speaking of the size of adits, the supposition is that only sufficient area is needed for transportation. In thick mineral deposits the area would not be appreciably increased, but in deposits up to 12 feet in width it would probably be made the width of the deposit. This last statement requires some qualification, for, if either wall is frozen, it may not be necessary to take the entire width for the vein, while, if either wall is separated by well-defined selvage, it will probably be necessary to excavate the entire width. In wide, large mineral deposits, the width of cars has some bearing on the width of the passages, particularly where machinery is the motive power. Such mines should be wide enough to permit a man to stand between the walls and the moving car.

11. Adit Tracks.—For the sake of economy, it is customary in most cases, to purchase light mine rails and place ties some distance between centers. Cars having a capacity of 1 ton and a weight of about 1,100 pounds should have a substantial track, with rails weighing at least 20 pounds per yard and ties spaced 2 feet between centers. The ties should have a 5-inch face and be 5 inches thick. The length of the ties will depend on the track gauge, but in all cases should project from 12 to 15 inches beyond the rails, to afford stability to the track. When rails smaller than 20 pounds per yard are used in mines, the ties must be less than 2 feet between centers, otherwise the tracks will bend and spread, and very often allow the car wheels to slip between the rails. To lift a loaded car on the track requires time and labor that should and can be avoided.

The grade given adit tracks should be in favor of the loaded car, as the empty car can be more readily pushed about. A good grade is about 6 inches in 100 feet, or a

½-per-cent. grade. Such a grade will make tramming easy and insure a flow of water in the ditches, provided they are kept clean.

12. Conditions Governing Tunneling.—In excavating tunnels, all kinds of rock conditions may be encountered: The solid, firm rock may be changed to weak, crumbling material. The rock that is firm and solid during excavation may be changed by air and moisture to what is known as *swelling ground*. Slippy rock or areas of rock that have been crushed by dynamic forces may be encountered, which will not stand without artificial support. Besides these conditions, there are others likely to arise, the most dreaded of which is quicksand, which involves both expense and trouble to excavate. Different conditions must be overcome by different systems of working and supporting the excavation; hence, much depends on those in charge of the work.

13. Locating Adits or Tunnels.—It is always advisable to locate the adit opening as near the water level as is consistent with mill or shipping facilities and the delivery of mine supplies. Cross-cut tunnels should not be driven at depths much exceeding the known depth of the vein, because of the uncertainty of ore deposits continuing in depth. If it is possible, cross-cut tunnels should be driven to meet the veins pitching toward them, and by this means decrease the length of the tunnels.

TUNNELING THROUGH LOOSE GROUND

14. Beginning a Tunnel.—Unless the tunnel is located in a narrow valley, or in a place where rocks are uncovered, it will be necessary to make an open cut in loose ground until a height somewhat greater than that of the opening is obtained. The thickness of the loose ground depends on the slope of the hill; in narrow valleys, it will be found that the soil is thicker on the south side than on the north side of the hill. If it is possible to reach solid rock by making an open cut, it is much better to do so, as otherwise the ground will cave above the entrance, as shown in Fig. 3.

In this case, a wooden portal was constructed and the ground above the adit was supported by timbers. In time, the timbers rotted and broke, letting the surface into the excavation, and necessitating the construction of a new portal. Owing to the fact that the water in running down the hill sank through the loose earth, the floor of the entrance was

Fig. 3

always wet and sloppy; hence, when a new portal was constructed, it was built of concrete and the masonry was carried back to solid rock. Besides the advantage of making a water-tight portal, concrete, made of six parts of broken stone $1\frac{1}{2}$ inches in diameter, two parts of clean sharp sand, and one part of Portland cement, will make a permanent lining and portal.

15. Fig. 4 shows the method usually adopted when a tunnel is begun in hard pan—that is, in material composed of rocks of various sizes and cemented with clay mixed with sand. Hard pan is difficult to pick and shovel when dry, and very difficult to excavate when wet, as it is then heavy, sticky, and liable to run. An excavation in hard pan is

carried on as an open cut until a height is obtained that will give the desired opening. The walls of the cut should be sloped, otherwise each rainstorm will cause a slide and fill up the ditch. The first two sets of timbers are placed back from the face, so that when the dirt assumes its natural slope it will not fall into the cut, but will cover the lagging above those two timber sets. The stones shown on the

FIG. 4

lagging above the timbers are intended to hold the timbers in place until the earth has sloped naturally and covered them. The method followed for excavating will depend on whether the ground is wet or dry. Sometimes the entire face may be excavated and a distance gained that will permit of the timbers being inserted without trouble; again, only a small portion of the face can be excavated at a time, and forepoling must be adopted.

16. **Placing Timber Sets.**—In soft ground, such as in the case under discussion, there must be a mud-sill *a*, Fig. 5, for the legs *b* to stand on, and the excavation requires full timber sets until solid rock is reached. The first two sets are held upright by the weight placed on the lagging *c* above them, and are tied together by the planks *d* nailed as shown. In this kind of ground, the sets are placed about 30 inches from center to center, and are backed by sawed lagging *c* above and with either split or sawed lagging *e* at the sides.

When the floor of the tunnel is composed of material that becomes soft and permits the sills *a* to sink into it, planks are laid lengthwise of the tunnel and the sills placed on them. This arrangement distributes the weight coming on the timbers over a wider space on the floor than that covered by the sill alone. The sill is first placed in position, then the legs are stood on the sill and tied by the planks to the adjoining set; then the caps are put in place. The lagging is next inserted, beginning at one side and continuing across to the other. The side lagging *e* is inserted behind the posts from the bottom upwards, each piece being sawed to a definite length, as shown by the dotted lines on the posts. As soon as two or three pieces of lagging are in position, filling is placed back of them to prevent them from being broken by any sudden rush of earth.

FIG. 5

17. Tunnelling Through Talus.—Rocks that have broken from cliffs and slid or rolled down the side of a steep slope and then become banked up are termed **talus.** If the descent has been gradual, the material is smaller in size and more uniformly compact than if the descent had been rapid. Talus may occur in pieces of all sizes, and, as it may be loosely held together by sand and clay, it is likely to move whenever disturbed by excavations. To penetrate such material, an excavation somewhat less than the intended size of the tunnel is driven 6 or 8 feet in advance of the proper-sized tunnel and timbered with a false

set *a*, Fig. 6. The object of the smaller tunnel is to prove the ground and ascertain whether there are large boulders or angular rocks that will require blasting, and to keep the material from running into the excavation, as it would if that were made full size. When removing a large stone, material of this description is liable to run and fill the excavation; hence, only enough space is excavated to place one false set at a time, with the timbers skin to skin.

A permanent set *b* is put in place by removing a false set as at *c*, and excavating a space only large enough for its insertion. To accomplish this, a small excavation above the false sets *a* is made and the cap *d* inserted. Next, the planks *e* are placed one at a time so as to form a cover above. When *d* and *e* are in place, the widening at *c* is comparatively easy, and then the permanent timbers *b* are placed. As soon as the permanent timbers have replaced all but two or three false sets, the smaller excavation is again advanced and the operation of enlarging the excavation repeated. This system of excavating and timbering is continued until solid rock is reached. It has been suggested that the method be followed when excavating wet hard pan, but there are other systems that have proved efficient in coping with such material.

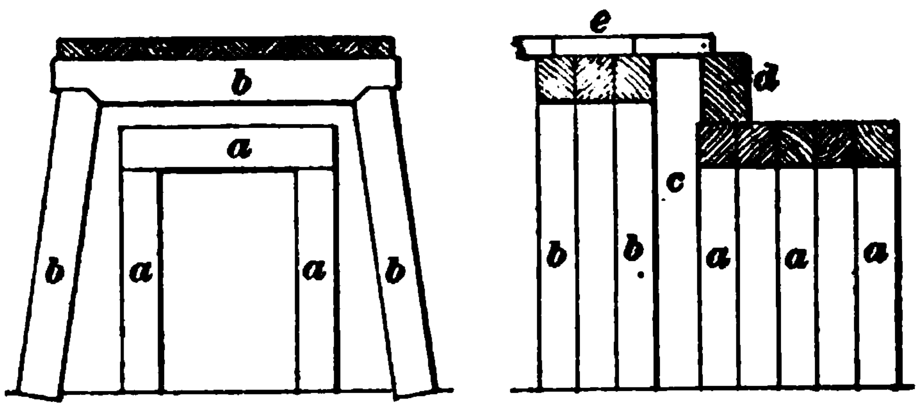


FIG. 6

18. Tunneling Wet Hard Pan.—When hard pan is very wet, it will run almost in a stream, the water seemingly converting the dirt into a liquid mud that carries along stones as it moves. It is very difficult to drive a tunnel through such ground on account of its mobility. When hard pan is moderately wet, it has the consistency of putty, and its heaviness and stickiness make it difficult to dig and handle; besides, as little reliance can be placed on its standing alone, even for a short time, it must be supported

by special timbering. When dealing with wet hard pan, slow and methodical work is required, and if arrangements can be made to drain the water away from the excavation it will be advisable to do so. When timbers are in place and the material is packed about them and drained, they will have little destructive pressure to withstand, since the material is capable of sustaining some of the pressure.

19. Forepoling.—The method generally adopted for penetrating loose or wet stuff is that known as **forepoling**, and the timbers for supporting the excavation are known as **bridged sets.** Fig. 7 shows an elevation of a bridged set of timbers, in which *a* are the legs; *b*, the cap, or collar; and *c*, the sill. The timber sets have, of course, to support the main part of the pressure coming on the excavation, yet the spiling driven into the ground through the spaces *h* between the bridge pieces *d* and *e* supports some of the pressure and transmits it to the other timbers. Spiling for this work is made of planks, usually 2 or 3 inches thick, which are given a chisel edge at one end in order that when driven with a maul they may penetrate the ground ahead of the excavation. These planks, which are from 3 to 5 feet long, permit a certain amount of digging to be done before the pressure bends them inwards, and if they have been driven in properly, an excavation may be made large enough for a set to be placed. Continued pressure bends the spiling inwards until eventually it is stopped by resting on the bridged sets, where it answers the purpose of lagging.

Fig. 7

20. Driving Spiling.—Fig. 8 is a plan showing the method of driving the spiling *j* into the ground between the

bridge pieces *d* and the legs *a* of Fig. 7. The figure further shows the tail-pieces *k* that keep the spiling pointed out and assist them to withstand the side pressure. The spiling is

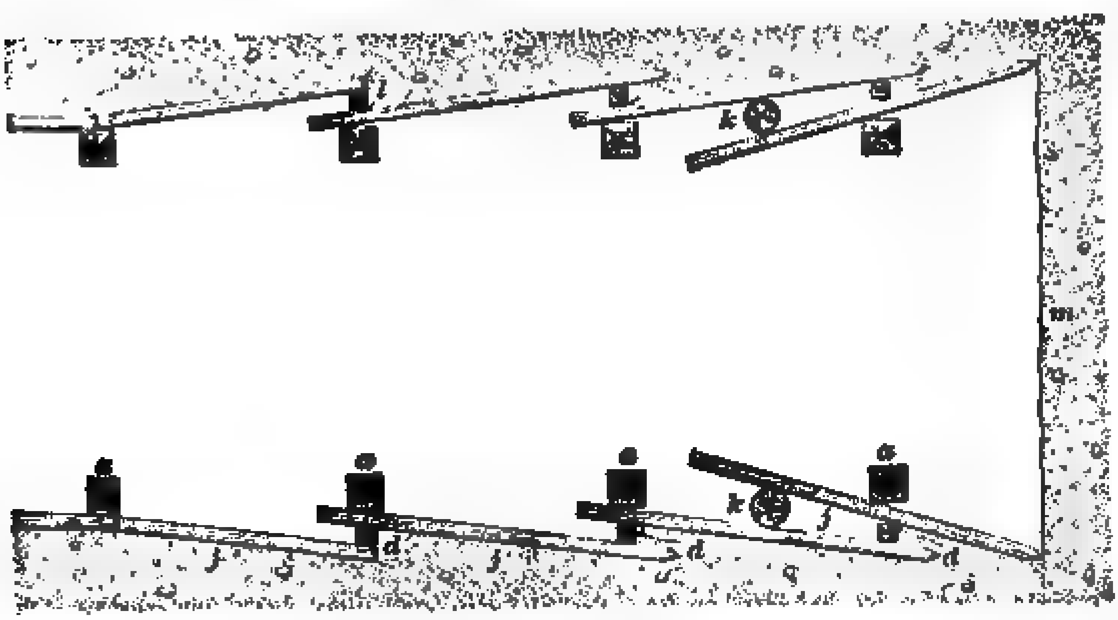


FIG. 8

driven in uniformly, each one a little at a time, in consecutive order about the bridged set, it having been found that this method of advance is more effective than where each spile is driven its full length at one time.

FIG. 9

Fig. 9 is a longitudinal section of the bridged set of Fig. 7, through an excavation where the roof is held in place by

spiling *j* while the face *m* is excavated. The legs and collars of the main timber set are shown at *a* and *b*, respectively; while at *e* are shown the collars of the bridge set. The spiling *j* is driven as before, with a maul, and is kept pointed properly by the tail-piece *n*.

21. Breast Boards.—When ground partakes of the nature of quicksand, the face, as well as the sides, roof, and floor, must be supported both during and after excavation. The face may be wedged or held back by breast boards. Fig. 10 is a longitudinal elevation showing the method of spiling and preventing the face from caving—or *running*, as it is termed—into the excavation. The breast boards *h* are advanced one at a time, beginning at the top and working down, by removing a small quantity of material. As soon as sufficient material has been scraped out, the breast board is inserted and braced back to the timber set by the sticks *i*. This method of work is continued until room has been obtained for a timber set *a*, *b*, *c*. It will be noted that the spiling plank *d* rests on the breast boards and that it is advanced regularly, as shown at *g*. This arrangement provides space for the bridge sets *e* on which the spiling eventually rests.

Fig. 10

In very loose ground it will be necessary to brace the timber set nearest the face back to the other sets, in order to prevent it from being moved out of plumb.

22. Side Brace Sets.—Fig. 11 shows one method of fitting in reinforcing side braces between two sets. The legs, cap, and sill of the regular set are denoted by *a*, *b*, and *c*,

respectively; while the legs, sill, and cap of the reinforcing set are shown at *d*, *e*, and *f*. Since the reinforcing sets are not mortised and tenoned, the braces *i* will prevent them from being pushed out of place. The braces *i* can then be held in place by legs *l* or posts *p*. Angle braces are not advisable in these timber sets, as the pressure would have a tendency to push out the legs of the timber set at the cap and sill joints. Again, angle braces are difficult to fit properly, and, as time is an object, the method shown for reinforcing sets is probably the better to follow, as it requires no jointing.

FIG. 11

23. Wedging Loose Ground.—In making excavations in loose ground that will not stand alone, every precaution should be taken to prevent it from running. Even with the greatest care, small caves cannot be prevented, so that the spiling should be advanced methodically and systematically. When such work is begun, it should be continued uninterruptedly until finished, to prevent any great change in the new conditions brought about by excavating the earth. If the ground is of so soft a nature that it will ooze in appreciable quantities through the spaces between the spiling, some

FIG. 12

other system than that of spiling should be adopted. The wedging system shown in Fig. 12 was invented in Europe and used successfully in several cases. The work of excavating is carried on in three stages. The wedges *a* are driven into the face as close together as they can be placed; the wedges *b* are next driven, and as they are advanced the floor wedges *c* are inserted and driven into place. The spiling *d* is kept in advance of the wedges *a*, *b*, and *c*, both above and at the sides of the excavation. As the wedges crowd back the quicksand, this stage of the work must be carried on systematically until the advance will permit a temporary set of timbers *h* to be put in place. The planks *g* are placed across the excavation, and lengthwise on them the stringers *f* on which the timber sets *h* and *e* rest. In case the pressure on the face becomes too great for the wedges to withstand, a few holes are bored through their centers and the pressure relieved by allowing the pent-up material to spurt through these holes into the excavation. As soon as the pressure is relieved, the holes are plugged up and the wedge driving is continued.

The temporary timber sets *h* must be so spaced that the regular sets *e* following will be skin to skin. Since quicksand is heavy and exerts a continual pressure, the bridged sets *h* and their spiling would support the excavation for but a short time—hence, the necessity of reinforcing them with the four-piece sets placed close together. Timbering of even this strength will not stand up if the quicksand is at a considerable depth below the deposit, so that arches of masonry must be constructed inside of them. The system of wedging is not applicable where the excavation is at a considerable depth beneath the surface, and other means are adopted to hold back the ground until it can be timbered.

Quicksand in either shaft sinking or tunneling operations is always a course of anxiety, and its excavation should not be attempted without a thorough knowledge of its character, and definite information regarding the successful methods that have been adopted to overcome its action. Sixteen

feet of quicksand has cost $16,000 to penetrate where greater engineering skill put the tunnel through for $1,000.

24. Driving a Tunnel Through Running Ground. The Croton aqueduct, which supplies New York City with water, is a tunnel for a considerable part of the distance traversed. At one point, between Tarrytown and Ardsley,

Fig. 13

near shaft 13A, a pot hole was encountered that contained wet running ground. A cross-section showing the relation between the country rock and the pot hole is given in Fig. 13. The country rock *a* is dolomitic limestone, and the pot hole *b*, which extended 110 feet along the tunnel line, contained hard, compressed, yellow mud, composed of fine particles of mica, sand, and clay. Through this mass were several strong water-bearing seams filled with sand and gravel.

The water from these seams formed with the mud a mixture of the consistency of pea soup. It was in effect the equivalent of quicksand. In the figure, *c* is shaft 13 driven to the tunnel *d*, and *e* is shaft 13A. As the tunnel was being advanced a cave occurred at *f*.

25. When the bad ground was first tapped, a mixture of decomposed limestone, clay, sand, and dirty water ran into the tunnel and partly filled it for a distance of 125 feet from the face. After 3 days, when the water became clear, the fissure was plugged with straw and the heading advanced 20 feet, when without warning another rush of mud much greater than the first drove away the workmen. Timbering of the form shown in Fig. 14 was then commenced at the soft rock, and by this means the heading was advanced 34 feet and seventeen sets of timbers placed. A bulkhead was driven at the end of the timbers for the purpose of holding back the soft ground; but for several days the mud was forced inwards with such pressure that oak logs 24 inches in diameter that were used as rakers at the bulkhead were crushed within 24 hours. The cave *f* at the surface now occurred, and being considered the mouth of the fissure, an abortive attempt was made to clog it up.

Fig. 14

Fig. 14 shows a bottom drift *a* that was started in the bad ground. It was 5 feet in the clear, and was advanced through 35 feet of decomposed limestone and 19 feet into the heavy ground that caused the timbers to crack. After 11 weeks' work this was abandoned, as the mud ran in as fast as it was

removed. The great trouble was to drain the ground, which furnished 160 gallons of water per minute.

26. After several experiments, the following system was introduced, and the work completed within 14 months without serious interruption: The lower drift *a*, Fig. 15, was driven 8 ft. × 6 ft. and timbered for a distance of 25 feet. This was widened out by placing the bearing bars *b* under the caps *a*, and supporting them on posts that rested on

Fig. 15

longitudinal sills *c*. The process of widening out was done by spiling a small excavation, and then placing the bearing bars *d*, the sills *e*, and the posts *f*. This excavation was continued and secured until the desired width was attained, after which the opposite side was widened out.

The widening having been completed, spiling 6 feet long was driven outside the sill *e*, and the enclosed spaces were excavated in pockets, the posts *f′* resting on foot-boards placed at intervals in the mud. Spiling *g*, Fig. 16, was then driven across the bottom of the excavation about 15 feet

ahead of the masonry h, to prevent the mud from being pumped out from beneath the timbering and masonry platform. Longitudinal sills i, Fig. 15, of variable lengths were then inserted in the excavated spaces between the posts f'. Underpinning j was next employed, in order to fix the longitudinal sill in the space interrupted by the line of posts. For this purpose, immediately behind the first post, Fig. 15,

FIG. 16

a cross-sill k 6 feet long was raised on blocks 6 inches above the sills in place. To this sill the weight was now transferred, and the first post was removed. The second post was removed by shifting the pressure in a similar manner, but the third post was supported by the underpinning j. The sill l was then slipped into place, as shown in Fig. 16, and the cross-sills k inserted. The posts f' on the outside of the excavation could not be removed without danger of damaging the structure; so that longitudinal pieces were placed

between them. The excavated space in front of the platform was used as a sump.

27. As soon as the platform was completed, the cross-beams *m*, Fig. 17, were placed under the bearing bars *b* and *d*, about 3 feet between centers. These beams were supported by inclined posts *n*, and were firmly wedged up to the bearing bars. The posts *f'*, Fig. 15, were removed, and the

FIG. 17

central portion of the masonry *o*, Fig. 17, constructed; then the remainder of the invert *o*, Fig. 18, together with 2 feet of the side wall and backing, was built in.

28. During the construction of the masonry, the top heading *p*, Figs. 17 and 18, was driven about 15 feet, and the crown bars *q* were put in place. The back ends of the bars rested on the arch already built, Fig. 19, while the

heading ends of the first two were supported by the posts resting on the cross-sill s and the two longitudinal sills t, placed in the top heading. The excavation was carried down and out, and the other bars were supported by the struts a resting on the bearing bars b and d, Fig. 17. The large beam v, Fig. 18, 24 in. × 24 in. in cross-section and 26 feet long, was next placed ahead of the masonry, and

FIG. 18

supported by posts w, Fig. 18, resting on sills x, Fig. 19, in the bottom, and strutted back by rakers z. Posts a', Fig. 18, were placed under the crown bars q and set on the beam v, which also served as a bearing for the bulkhead against the mud in the face. When the section was completed, the sill was removed, the ends being built in. To support and strengthen the crown bars q, the segmental timbers b' supported by posts resting on foot-blocks were put in, at

distances of 1 or 2 feet apart as deemed necessary. The side walls and the arch were next constructed. No attempt was

Fig. 19

made to withdraw the crown bars, which rested on the masonry arch when that was built.

TUNNEL PORTALS

29. Stone Tunnel Portal.—When the portal of a tunnel is to be permanent, it will be found cheaper in the end to construct it of stone or masonry. Usually, such portals are given an arch whose radius has a length that will give a fair curvature but will not be too flat. Several designs of portals are illustrated in Fig. 20. The portals shown at (*a*) and (*b*) are partly wood and partly dry masonry. While they have a neat appearance, they are no more permanent than an all-wood portal and have the same disadvantages. Fig. 20 (*c*) is a single-track tunnel portal with the crown

stones *a* dressed to form an arch and fit snugly to one another and the skewbacks *b*. Such masonry is carried back until the rock walls of the tunnel are solid. Fig. 20 (*d*) is a flat-arched tunnel portal for three tracks. The keystone *a*, crown stones *b*, skewbacks *c*, and the lining walls must be

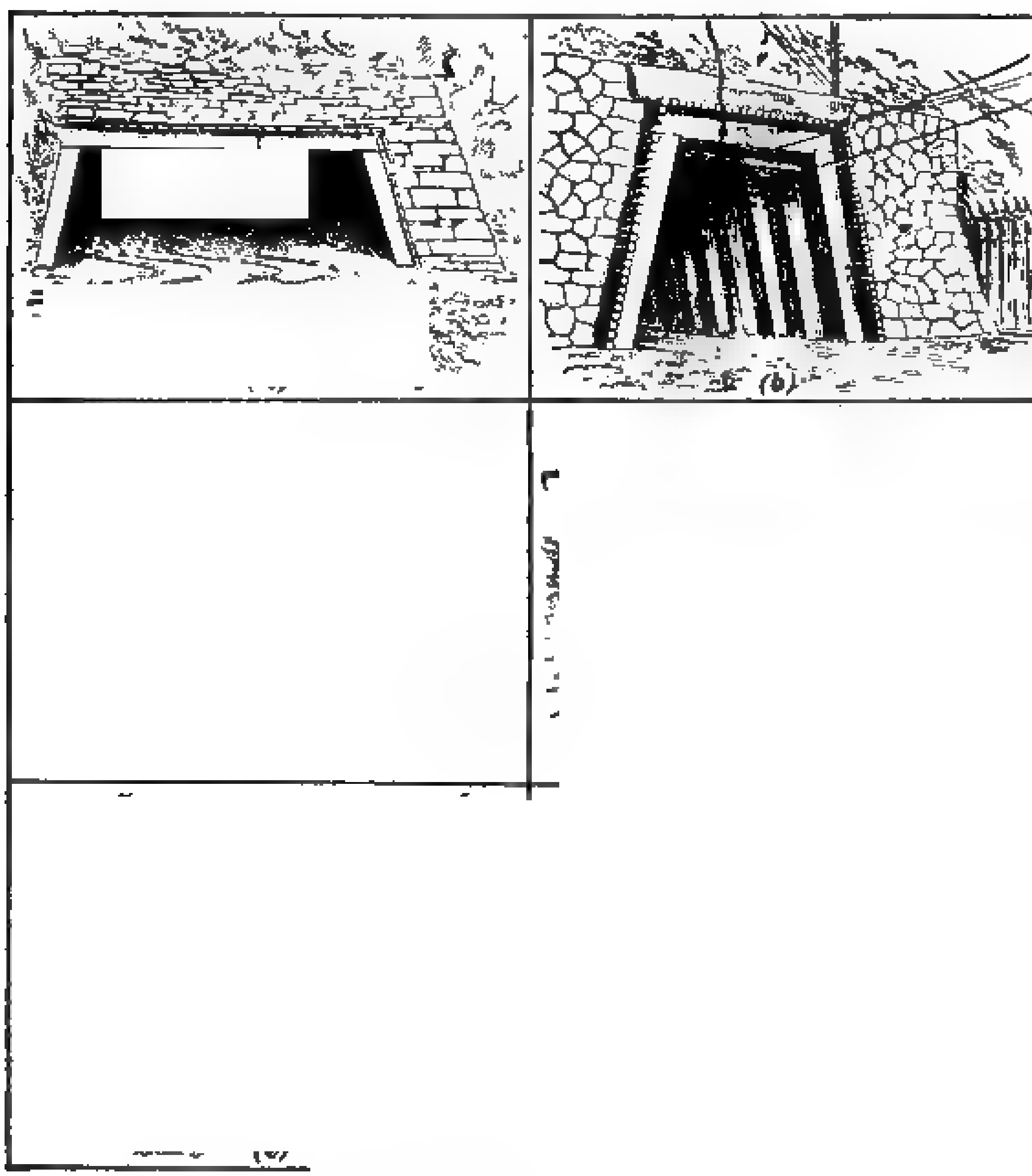


Fig. 20

dressed and laid with more care than those in Fig. 20 (*c*), on account of the greater thrust on the skewbacks due to the flatness of the arch. Fig. 20 (*e*) does not differ materially from Fig. 20 (*c*), except that it has dressed-stone side walls has no skewbacks, and has a more rounded arch.

30. Concrete Portals.—Fig. 20 (*f*) is a concrete arch for two tracks. There is no necessity for constructing the arch of concrete blocks, as illustrated, for concrete and expanded metal are just as serviceable and as strong as concrete blocks. The concrete is carried out as wings on each side of the portal to prevent earth or rock from falling on the track or clogging the ditch. The entrance to a tunnel is nearly always wet in a temperate climate, unless means are employed to prevent the entrance of rain and surface water from above. If the lining of portals is carried into the tunnel until solid rock is reached, the water will take some other course and will not drip into the tunnel. Concrete linings at portal entrances keep the entrance dry and are cheaper to construct than stone portals.

TUNNEL LININGS

31. Concrete Tunnel Lining.—A novel and successful method of tunneling was practiced in connection with a system of subways excavated beneath the city of Chicago. The subways were excavated in a stiff clay that contained practically no water and would stand without caving or swelling. Compressed air at a pressure of 23.5 pounds per square inch was kept on the face, not because it was needed, but as a precautionary measure in case the work would have to be left at any time unfinished or unlined. The clay was gouged out with a tool similar to a carpenters' draw knife, to widths varying from $7\frac{2}{3}$ to $14\frac{5}{12}$ feet and to heights varying from $11\frac{1}{6}$ to $15\frac{2}{3}$ feet. The concrete lining *a*, Fig. 21, was made 20 inches thick, was rammed behind board templets *b*, backed by steel centers *c*, and held in this position until it had thoroughly set. The centers and boards were then removed and used again. There were three 8-hour shifts of twenty men each at this work. Two of the shifts did the cutting and mucking, while the third shift set the centers and rammed in the concrete from the bottom upwards. By this means, it was possible to keep the rails close up to the face and remove the clay in cars almost as

fast as it was cut. With a force of 1,450 men and several pairs of headings working toward each other, 12 miles of tunnel were completed in 10½ months. The advance in each heading was on an average of 21 feet per day.

Fig. 21

32. Brick Tunnel Linings.—In quicksand or ground such as that encountered in the Croton aqueduct at shaft 13A, which went to pieces the moment water came in contact with it, the tunnel is ordinarily lined with bricks. The lining shown in Fig. 22 is composed of three arches, the

invert *a*, the side arches *b* having the same spring, and the top arch *c*. The invert is laid on a bed of sand and is built to a center. The practice of building flat floors for tunnels in soft ground has been discarded, as it has been proved that the form could not withstand pressure nearly so well as an arch. The first arched form adopted was elliptic, but the amount of space lost was too great, unless a ditch was desired beneath the track. The inverted arch was next

FIG. 22

tried, in order to save space and at the same time furnish sufficient strength to withstand the pressure coming from below, as in such ground pressure comes with equal force from all sides. The inverts were so successful that they are now generally adopted both in soft ground and in soft rock.

33. Backing for Tunnel Masonry.—Where bricks—or, in fact, any masonry—are employed for lining tunnels, the space between the masonry and the walls of the excavation should be filled up. The excavation is necessarily made of

greater area than the finished tunnel in order to accommodate the masonry, but the size should be approximated and as little extra excavation made as is consistent with good work. It is not always possible to obtain just the area needed, in which case, as shown in Fig. 22, the space *d* back of the lining and inside the timbers should be filled in with concrete. These remarks apply equally well to excavations in quicksand, it being always borne in mind that, where masonry is used, no spaces should appear between the walls. Furthermore, all timbers needed for temporary support should be removed and the spaces filled with some material such as sand or clay. Since the backing receives the pressure that comes on the masonry, its construction should be such as to distribute the weight uniformly and under compression act as a resisting material. Sand answers the requirements admirably; concrete, not having the same flexibility, is not considered so good for the purpose, especially where rock walls surround the lining.

34. Arch Construction.—Two methods are in use for constructing arches to a given form and size. In one method, a sweep or radius stick is employed and in the other a center or wooden templet. The sweep is an ordinary straight stick of a specified length, which is rotated about a point central to the span or throw of the arch. As it is a rather difficult matter to locate this center and keep it central—especially where men have to work on platforms—arches are more readily and accurately constructed by employing the method of centers. These templets of lumber are cut and arranged to furnish the desired curvature when framed and fastened together. The frames are carefully set in position about 12 or 18 inches apart, and the brickwork is built back of them. When the centers are properly placed, the mason has no excuse for making mistakes, such as has happened when sweeps were used. When inverts are constructed, two sets of centers are needed: one for the inside and one for the outside of the invert, since each has a different radius. The bottom center

is used to furnish the proper curvature for the sand or cement bed on which the masonry rests, after which it is withdrawn. The top center is then put in place, and the wall built up to it.

ROCK TUNNELING

35. Bench Tunneling.—Rock tunnels of small area can be driven in full section; tunnels 12 ft. × 12 ft. and upwards in section should preferably be driven in benches. Fig. 23 shows a front and side elevation of a tunnel driven in benches. The upper bench *a* is kept in advance of the lower bench *b*, and in a 12′ × 12′ tunnel would probably have a height of 7 feet. Bench tunneling is not appreciated as fully as it should be from an economical standpoint and is practiced only when the area of the

Fig. 23

tunnel demands that this system be followed. It has, however, some features worthy of consideration. The reduction in the number of drill holes, compared with the full-face method, effects an economy in drilling and blasting, which are the most expensive part of mining. As the cost of explosives increases in proportion to their strength, bench tunneling means a decrease in the consumption of high explosives for center cuts. Then, too, as the bench is never broken close to the face, this method makes loading easier and thus saves labor; furthermore, the work can be advanced as quickly by this as by the full-face method.

36. Drilling Blast Holes in Tunnels.—Where a 6′ × 7′ or a 7′ × 8′ tunnel is driven in mines by hand, it is customary to let the work out on contract for so much per running foot. The miners that drive such tunnels make a little more than day wages, so that the work is generally performed more expeditiously in this manner than when done in company time. The average cost of this work is about $8 per linear foot. In hand drilling in such confined places, the miners usually work in gangs of two and three. Where two are in a gang, 1¼-inch bits are used, and the holes are put in to a depth of about 36 inches. When three are in a gang, the holes are bored to a depth of about 42 inches. The size of the bit used has much to do with the rate of drilling; for example, the quantities of rock pulverized are as the square of the diameters of the hole—in other words, the width of the bits. If, therefore, a 1-inch bit is used, 14 feet should be drilled, compared with 9 feet drilled with a 1¼-inch bit. This has considerable bearing on tunnel driving as well as on mining, for, if the long center-cut holes are driven with a 1¼-inch bit and high explosives are used, the shorter side holes can be driven with a smaller bit, and, with the use of high explosives, perform the desired work. It is customary, when using machine drills, to make the center holes with a larger diameter than the side holes, and to use high explosives when the ground is tight. The system here advocated for hand drills differs only in the use of high explosives for the side holes.

37. Tunnel Sections.—The smallest tunnel section that can be worked to advantage has an area of 6 ft. × 6 ft.; below this the small area gives the men trouble, while above it they can work better. There is no advantage to be gained by making a tunnel 8 ft. × 10 ft. in area when one 6 ft. × 8 ft. in area will answer the requirements; in other words, the size of the excavation should be limited to the actual needs of the mine, it being borne in mind that too small a sectional area is not economical. The width of tunnels should be such that a man can pass between the walls and a moving car without injury.

When a tunnel is driven in rock having thick layers produced by sedimentation or other causes, the section should be made smaller at the roof than at the floor. This is illustrated in Fig. 23, which shows that such rock when arched is able to support the roof without artificial aid. Tunnels may be given vertical side walls and parallel roof and floor walls when driven in solid compact rock free from fissures and bedding planes. The walls of tunnels driven in shale and slaty rock must be supported by timbers or masonry.

FIG. 24

38. Placing Shot Holes.—The American method of tunnel driving is to do all drilling, firing, and mucking separately and to arrange the drill holes so that they may prove most effective when fired in rounds.

Fig. 24 shows the face of a tunnel and the direction given the holes. The holes marked *a* are usually driven in pairs at an angle, so that their inner ends will come nearly together. These center-cut holes are made about 10 feet long, and are loaded with high explosives containing about 60 per cent. of nitroglycerine or a powder having an equivalent strength. The side-cut holes *b* are made about $8\frac{1}{2}$ feet deep, are charged with 40 or 50 per cent. of

FIG. [illegible]

nitroglycerine powder, and are fired after the center-cut holes. Fig. 25 is the plan of a heading with the holes in position, and shows that the center-cut holes *a* must necessarily be longer than the side-cut holes *b*, on account of the angles given them, and also because they must extend beyond the side holes in order that the latter may have two free faces to work on.

FIG. 26

Fig. 26 gives a longitudinal section through the center-cut holes *a*, Fig. 24, showing their inclinations relative to the face. It will be noticed that the top holes go to the roof line, while the bottom holes go slightly below the floor line. This arrangement will keep the area of the tunnel nearly uniform in compact rock; but in seamy rock it cannot always be relied on to do so. Fig. 27 shows in longitudinal section the position given to the side-cut holes. The center hole is made the longest because it is to break a larger area of ground, which it can do because the resistance that the explosive has to play against is not so great as for the upper or lower holes.

FIG. 27

39. American Center-Cut System.—The American center-cut system adopted by the Newhouse tunnel engineers in Colorado is shown in Fig. 28. The arrangement of the holes is indicated by the arrows, but is peculiar at the upper

center part of the tunnel. It will be observed that the back, or upper, holes *a* point upwards, and that a plunger hole *b* points downwards. The center-cut holes *c* point inwards, and the side holes *d* point outwards and upwards. The holes *c* are fired first, and then the side and top holes.

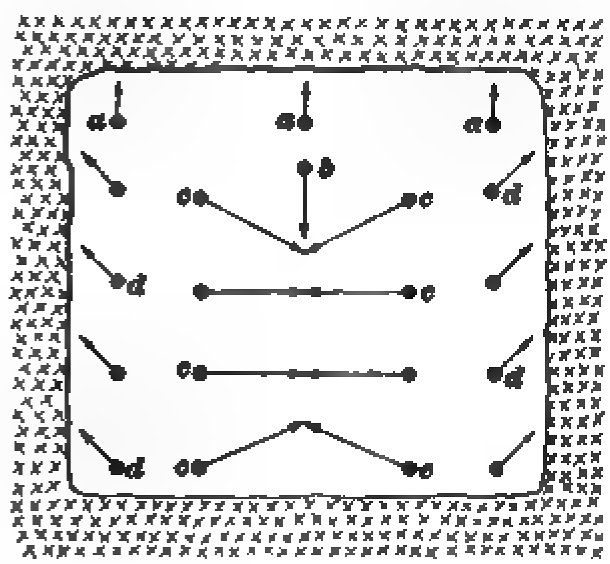
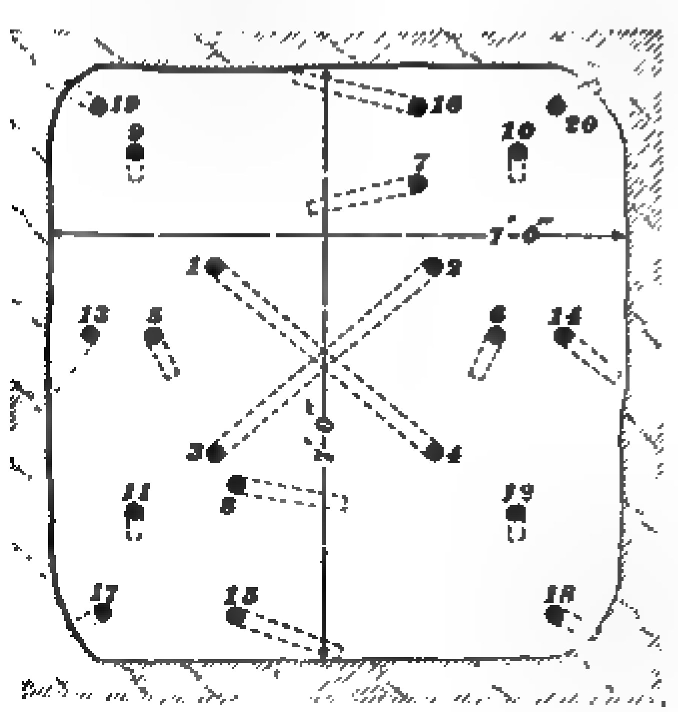


FIG. 28

Another system of placing holes so as to remove a center cut is shown in Fig. 29 (*a*) and (*b*). In this case the four holes *1*, *2*, *3*, and *4* are drilled toward a common center, loaded, and fired first. The holes *5*, *6*, *7*, and *8* are loaded and fired next in order; then the remaining holes are loaded and fired in one volley. This method of loading and firing may, of course, be varied to



(*a*) (*b*)

FIG. 29

firing consecutive rounds in holes *9*, *10*, *11*, and *12*; *13*, *14*, *15*, and *16*; and *17*, *18*, *19*, and *20*. The breaking-in holes *1*, *2*, *3*, and *4* must be fired first, no matter in what order the others

are fired; if, however, the holes are fired in the first order given, it is probable that satisfactory results will be obtained, and certain that a saving in time will be effected. It should be understood that all the holes must be drilled before any are loaded and fired. Many accidents have resulted from drilling and firing in rounds, the drillers having supposed that all the holes were exploded, when such was not the case.

40. Tunnel Grade Lines.—One difficulty experienced in driving headings is to keep the floor at or about the grade line. This seems invariably to creep up 1 or 2 feet and of course the roof follows the grade line. Miners claim that this trouble is due to drainage, or to bad rock that breaks high, while as a matter of fact it will be found that the drill holes were not extended far enough, as practice has demonstrated that ordinary rock will break as desired by placing the drill holes within a foot or less of the line. If, after making an excavation, the floor must be blasted to keep the grade, tunnel driving is an exceedingly expensive operation. The size of the tunnel can be kept more uniform by the bench system than by the other system. The areas of the heading and bench should be so proportioned that the area of the top bench will be sufficient to permit the men to work satisfactorily; at the same time, if it is possible to do so without interfering in this particular, the cross-sectional area should be made smaller in proportion to the lower bench in order to permit of the economical use of powder.

41. Newhouse Tunnel.—The Newhouse tunnel starts at Idaho Springs, in Clear Creek County, Colorado, at an altitude of 7,543 feet. It is a cross-cut tunnel having a cross-section 12 ft. × 12 ft. in area, and is intended to penetrate a mountain 5 miles and terminate under Quartz Hill, at Central City, 2,000 feet below the surface. In 1 year, 2,925 feet of this tunnel was driven with air drills at a cost of $21.45 per running foot, excluding the cost of permanent track and equipment. The method of carrying on the work is systematic, modern methods being adopted for power and haulage.

42. The drilling crew consists of five men, who begin work at 7 A. M. and continue until drilling is finished, usually about 5 P. M. These men remain at work at the face of the tunnel until they have put in their round of holes, when they leave the drift and are succeeded by the powder gang. The powder gang, which is made up of five men, takes down the drilling machines and removes them to a safe place, and then charges and fires the first round of holes; these duties take up about 40 minutes. There are usually five rounds to be loaded and fired, occupying the men about 3½ hours. After a charge has been fired, compressed air is used to drive out the powder smoke, after which the powder gang cleans up the track and loads the first train of ten cars.

43. At 10 P. M., the mucking shift goes on and remains at work until the broken rock is removed and the face is ready for the drillers. The mucking gang consists of six men; the foreman does the picking and three men shovel while the other two rest. This is necessary because the men are obliged to work fast and load about 100 tons of rock besides cleaning up the tunnel. Quite often sledges have to be used to break up the rock to a size that may be loaded readily. The cars have a capacity of 35 cubic feet or about 2.5 tons, and between forty-five and sixty cars are loaded during the shift.

As shown in the following article, the total cost per foot for driving a portion of this tunnel was $28.74, although, if it had been driven in connection with regular mining work, the fixed charges would have been materially reduced. For instance, salaries and office expenses added to those items under miscellaneous amount to $3 per running foot. In addition to the items quoted, which is about the same as the cost for drill crews, premiums were allotted in order to stimulate the men to perform more work in a given time. This item alone amounts to $1.87 per running foot and makes a further reduction for the legitimate cost of tunnel driving. On investigating other items, it will be found that economical methods were in a considerable measure discarded for the sake of rapid tunnel driving.

44. The cost of explosives, track material, repairs, etc. is high, for the reason that, in order to put in their time, four men of the blasting gang were obliged to either lay track or set timber; moreover, no tamping was used when charging the holes. The cost of the dynamite was about 38 cents per ton of rock broken. The cut holes were charged with dynamite containing 60 per cent. of nitroglycerine, while the side and back holes were charged with 40-per-cent. dynamite.

Items	Cost Per Foot
Drill crews and foremen	$3.01
Trammers, blasters, and drivers	4.34
Blacksmithing	1.03
Engines	1.15
Ammunition	4.45
Oil and waste	.12
Coal	3.91
Mules	.32
Drill repairs	.82
Premiums	1.87
Tools	.33
Timber	.16
Track and material	1.84
Tracklaying and repairs	2.07
Engineering and surveying	.22
Salaries and office expenses	1.97
Miscellaneous, sundries	
Legal, insurance, and taxes	1.03
Total	$28.74

45. Tunneling on Croton Aqueduct.—The data from the Newhouse tunnel compared with the data supplied by shaft 15 of the Croton aqueduct furnish material whereby a practical idea may be obtained of the cost of tunnel work in the East and the West. The rock at the Croton aqueduct was hard compact gneiss with occasional seams of quartz, very similar to that worked in the Newhouse tunnel. The area of the heading was 145.5 square feet, while the Newhouse tunnel had an area of 144 square feet. Both managements used 60-per-cent. dynamite for center-cut holes, and

40-per-cent. dynamite for side holes. The number of holes drilled was the same, although differently arranged: on the aqueduct there were eight center-cut holes 10 feet deep, and twelve side holes 8 feet deep; while in the Newhouse tunnel there were ten holes drilled to a depth of 10 feet and ten drilled to a depth of 8 feet. The aqueduct tunnel was driven in two 10-hour shifts, and the Newhouse tunnel in three 8-hour shifts. Ordinarily, the 10-hour shift was divided as follows:

Operations	Hours
Mucking from 7 to 9:30	$2\frac{1}{2}$
Drilling from 9:30 to 4:30	6
Charging holes from 4:30 to 5	$\frac{1}{2}$
Firing from 5 to 6	1
Total	10

Smoke was driven from the aqueduct tunnel during the hour between shifts, so that no time was lost. The mucking was accomplished by ten men in $2\frac{1}{2}$ hours, while in the Newhouse tunnel it took six men 8 hours. The total number of men employed in mining per shift on the aqueduct was nineteen, and on the Newhouse tunnel sixteen.

46. In Table I is given a comparison between two- and three-shift work, as well as a difference in cost between tunnel work in the East and the West.

TABLE I

COST PER RUNNING FOOT FOR 12′ × 12′ TUNNEL

Work and Materials	Newhouse Tunnel	Croton Aqueduct
Labor in mining and handling material	\$ 9.74	\$ 6.47
Engine and shop labor	2.34	.69
Powder	4.55	3.55
Oil and waste	.12	.12
Coal	3.91	.17
Salaries	1.97	1.46
Total cost	\$22.63	\$12.46

47. Cost of Driving Tunnels.—Tunnel driving is usually done on contract at so much per running foot for a cross-sectional area of 6 ft. × 7 ft. The price varies in localities according to the character of the rock and the cost of labor. In ore and rock or rock alone without timber, the average cost at eighteen mines in Colorado is $8 a foot, the lowest being $4 and the highest $12. This is for hand drilling, the wages of miners being $3 a day and the shifts 8 hours. Double-track adits cost about $25 a foot when driven by hand, the cross-sectional area being 7 ft. × 12 ft. The expense connected with driving an ordinary cross-cut tunnel either by hand or with machine drills is approximately the same; the main advantage in using machines is that the excavation can be advanced more rapidly, which is sometimes an item of such importance that the progress more than compensates for the cost of the drilling. Soft rock is not much, if any, cheaper to excavate than hard rock, on account of the heavier timbering required and the consequent delay in the work; moreover, there is usually more inconvenience from water. Quicksand or running ground is the most expensive material to penetrate.

The cost of driving the Nelson tunnel, now called the Wooster tunnel, at Creede, Colorado, was approximately $40 per foot. A long tunnel at Aspen, Colorado, driven in shale and Carboniferous limestone cost on an average $19 per foot for its 2½ miles of length; the labor on the last 6,000 feet, however, cost $11 per foot, timbering, compressed air, track, and transportation, with other charges, making up the balance.

48. Tunneling in Wet Formations.—When a tunnel is being driven through material containing much water, it is sometimes best to allow time for the water to drain out of the formation before making an advance cut. This does not always prove effective, but it shows what may be depended on in the way of water; besides, if effective, it will prevent a sudden rush of soft material into the excavation. When the rocks are known to contain pockets of water, it is

advisable to keep drill holes in advance of the work, and if old workings are being approached the holes should be driven both in the face and in the sides of the tunnel. Much damage has resulted and many lives have been lost through the neglect of this precaution. The depth of such proving holes should be about 12 feet; if the depth of the holes is less than this, they are possibly better than none at all, but water under great pressure, if confined by a wall of rock that is not more than 4 feet thick, is quite liable to break through into the workings.

SHAFTS

INCLINES OR SLOPES

49. Advantages of Inclined Shafts.—Since any opening into a mine other than a tunnel must have an inclination, the ore miner talks of inclined and vertical shafts. Inclined shafts that follow the dip of the deposit may vary in their inclination, for which reason they are not so economical nor so desirable for purposes of hoisting as vertical shafts. The advantages of an incline are that it follows the ore; that it eliminates the expense connected with cross-cut level driving and shaft sinking in barren rock; that the material excavated may sometimes pay for the cost of sinking; and, lastly, that self-dumping arrangements can be maintained at the top and give much better results than at the head of a shaft.

50. Disadvantages of Inclined Shafts.—For economical work, it is necessary that a slope be given a uniform grade as far as practicable. In order to do this, it will be necessary in most ore deposits to blast up the foot-wall in one place, take down the hanging wall in another, and in some instances bridge, with timbers, inequalities in the foot-wall. It is not unusual for inclined deposits to start with one inclination and then, with depth, assume another, so that the hoisting rope will in one place rub on rollers in the foot-wall and in another place rub on rollers in the hanging wall.

Such changes of inclination interfere greatly with the speed of hoisting and with keeping the skip on the track.

The pump discharge and steam pipes often cause considerable annoyance in slopes, since, if there is a depression, a bridge must be constructed for them to rest on; or, if there is a sudden change in the dip of the deposit, they must be fitted with an elbow or bend, and in some situations require special arrangements for their support. If for any reason the pump pipes are to be replaced, the work is more expensive and dangerous than in vertical shafts, on account of the difficulty encountered in connecting the pipes and anchoring them in the proper place. Inclines cannot be so readily drained when wet, nor driven to so great an advantage when dry, as shafts.

There is more or less difficulty connected with laying tracks on inclines, and in keeping the tracks when laid from creeping down the slope; since tracks are not used in shafts, this item is avoided entirely. Although there are a number of serious objections to slopes, nevertheless there are cases where they are used in preference to shafts, particularly where the amount of money to be expended is limited. For highly inclined shafts, skip cages are now constructed that greatly reduce the inconveniences encountered where wheel skips are used.

51. Location of a Slope.—The location for an incline should be selected with a view of obtaining the largest amount of ore from levels to be driven on either side of the excavation; but at the same time the contour of the surface should not be neglected. If it is possible, without making other matters too inconvenient, the opening should be made in the center of the property, provided there is a suitable dumping ground and no extensive depression at this point. In a matter of this kind, it is not always a good policy to locate the opening simply to make mining convenient; hence, the location of the buildings as well as the transportation facilities should be considered.

In some cases possibly, it may be thought advisable to drive the incline in the country rock rather than in the ore.

The foot-wall will be better for the purpose than the hanging wall, owing to the fact that ore will come by gravity to the loading stations. There are other reasons, however: the slope would be kept uniform and would not be thrown out of alinement by caving.

Fig. 30 illustrates a vein dipping into a mountain in such a manner that a cross-cut tunnel *a* would afford an opportunity for exploiting only a small area of the deposit. If a shaft were sunk at *b*, it would have to go a considerable depth before it cut the vein, and cross-cut levels *c* would have to pass through a large amount of barren rock if quick returns

Fig. 30

were needed. These levels would become shorter with depth, but after the shaft had passed through the ore they would increase in length as the depth of the shaft increased. All ore from such a shaft would have to be hoisted much higher than the outcrop, and while the adit *a* might be driven to drain the upper part of the vein, yet if the adit were not driven, the water would have to be pumped to the shaft mouth. The ore having been hoisted to the surface, it might be necessary to transport it to the valley for milling or shipping.

If the shaft *d* were sunk on the foot-wall side of the vein, the cross-cut levels *e* driven to intersect the vein would

become longer with each sinking, and being in unproductive rock would increase the fixed charges in mining. This expense added to the cost of mining would eventually eat up the profits; or, even if it did not, no valid excuse could be offered for adopting such a plan. It would seem that the proper method to follow in deposits of this kind would be to sink an incline. If this incline were in mineral, it would in some cases pay the expense of driving, or at least, quick returns would be obtained by driving levels in the ore. Eventually, a shaft may be sunk when the incline has reached a depth where it is known that charges can thereby be materially decreased, and that there is sufficient ore to be raised to more than pay for the cost of sinking. The incline can then be abandoned, or else used to remove ore mined from levels above the point where the shaft intersects the vein.

52. Size of Inclined Shafts.—The factors that determine the cross-sectional area of inclines are the expected output from the mine, the inclination of the deposit, and the strength of the hanging wall. If it is desired to hoist in balance and raise more than 200 tons daily, and the roof rock and inclination will permit, a double-tracked incline will be desirable. The thickness of the deposit may influence the height of the incline and consequently its area. Where the deposits come away from the hanging wall, or there is a selvage parting, the deposit must be removed to the hanging wall, if it is not thicker than 10 feet; but if its thickness is greater than this, it will be better to sink in the deposit, provided the material will hold together when supported by timbers placed skin to skin.

53. Flat Inclines.—Slopes having an inclination less than 15° may be termed **flat inclines.** Such inclines at ore mines are not made much over 12 feet wide even when the roof is good, although in some cases the width could be increased to advantage, without timbering closely. There is an excellent top rock in some of the iron-ore mines in New Jersey and New York that are worked through inclines having a width of from 12 to 16 feet. This fact, however, does not

alter conditions sufficiently to advise an increase in the width of the incline. If 12 feet is sufficient width for a slope, it should not be made wider, even if timber is not required. Where several lifts are worked simultaneously, a double-track slope is better than a single-track slope; for with the latter only a comparatively small output can be had, unless several cars can be hoisted at a time.

54. Hoisting Cars for Flat Inclines.—Mine cars can be employed on flat slopes for hoisting, provided the cars are constructed with this object in view. When such are used, the back end is made higher than the front end, and the cars are ironed and reinforced in such a way as to prevent them from being racked by the strains that come on them when starting.

Another system of hoisting is to use a slope car *a*, Fig. 31, on which the ordinary mine car *b* is run and fastened. The slope car is attached to the hoisting rope *c*, and having been loaded with a car, is pulled up out of the mine. At the surface, the loaded car is pushed off the slope car on to a track, and an empty car substituted. One advantage presented by the slope car is that it may be stopped and loaded at any level without the necessity of constructing an expensive loading plant. Slope cars are sometimes made with double decks, so that two cars can be carried at one time.

55. Hoisting Cars for Steep Inclines.—Inclines greater than 15° from the horizontal are considered steep,

as a man cannot, without considerable difficulty, travel up or down unless he has stairs or hand rails to assist him. Since any slight jolt from unevenness in the tracks or from flat wheels might shake material from mine cars running on

Fig. 32

steep inclines, skips on wheels are used until the inclination from the horizontal reaches 70°. In a shaft 300 feet deep on steep uniform inclinations, the friction of the rope on the rollers does not absorb much power, hence skips hoisting

out of balance or in balance are used to advantage. In case the inclination were not uniform, skips would be used; but there would be more friction and more power absorbed, and hence more wear and annoyance.

When the angle of inclination is greater than 70° from the horizontal, a skip on wheels will leave the track unless care is practiced in hoisting. In such cases it is customary to use guide rails or discard the skip entirely and substitute a slope cage. The slope cage is shown in Fig. 32 to be an adjustable iron or steel frame provided with wheels *a*, cage shoes *b*, safety dogs *c*, attached to and worked by coil springs *d*, and a hood *e*. The wheels run on iron tracks that are fastened to wooden cage guides not shown. The platform *f* is adjustable, while the frame *g* holds the car on the platform.

An objection to this slope cage is that the cars must be pushed on and off the platform from the end, thus making landing plats necessary at each level. The commendable features of the cage are the safety arrangements with which it is equipped, which prevent it from falling in case the rope breaks; its comparative lightness; and its ability to bring loaded cars out of the mine. Without going into details, it is stated that very frequently a saving can be effected where cars can be hoisted bodily from the mine and not dumped before the ore has reached its final destination. Where skips are used, the ore is loaded into mine cars, which are pushed to the loading station and dumped into the skip. The skip is hoisted to the surface and dumped automatically into an ore bin. The cars running to the mill or shipping station are loaded from the bin, hauled to their destination, and there dumped. By loading and assorting the ore in the mine and loading in cars at the face, two dumps and two loading stations may be dispensed with. This is not always practicable, however, even when slope cars and cages are employed; for the ore may need assorting and require another handling.

LOADING STATIONS

56. Loading Stations in Flat Pitching Deposits. On slightly inclined slopes the cars may be transferred from the slope track to level tracks by means of hinged drawbridges, which are lowered to the slope track. These bridges have tracks so arranged that cars coming down the slope can be run directly to the level track when the drawbridge is down, and loaded cars can be hauled from the landing to the slope track. When the bridge is up, loaded or empty cars are not interfered with by the bridge. The system is not very satisfactory, as considerable timbering is required to support the bridge and excavating is necessary to make a landing stage in the hanging wall. The slope car is preferable to this method of transferring cars from the slope to the level. At the lowest level in such mines, there is a turntable so located that the cars can be pointed up the slope for hoisting or toward the levels for loading. In some cases, skips are used with turntables instead of cars. When this is the case, the skips are unfastened from the hoisting rope, turned, and pushed to the stope for loading. As there is a loaded skip always ready to be transferred to the slope, no time is lost in hoisting.

57. Loading Stations in Steep Pitching Deposits. When the pitch of a slope is above 45°, skip turntables can be used on the lowest level of a mine, but they are neither satisfactory nor safe. The turntable must necessarily be horizontal, thus making an angle with the slope track. When the car starts, there is a quick jerk that is intensified by a sudden change from a horizontal to an inclined direction. These jerks have a tendency to disturb the ore in open skips, so that some of it may be dislodged farther up the slope and roll down to the landing, thereby causing damage or injury. Hence, on such slopes a closed skip must be employed, the loading of which at the face, while possible, is both expensive and awkward. When, therefore, the pitch is steep, covered skips are used, and this necessitates the

construction of a pocket at the lowest level into which the skip is lowered. The mouth of the skip is just a little below the level, so that mine cars can be dumped into it. At upper levels, loading stations are constructed, and are equipped with aprons to direct the ore from the car directly to the mouth of the skip, or with pockets from which the skip is loaded. In Fig. 33 is shown a landing at an upper

Fig. 33

level. The mine track *a* is laid to the hopper *b* in order that mine cars can be dumped directly through the hopper into a skip beneath. At *c* is shown the skip track, while the pipe *d* is used for purposes of ventilation.

Fig. 34 shows an arrangement for dumping direct from the mine car into the skip. The slope in this case has a moderate pitch with strong walls. The timbering consists of stulls *a* without caps or lagging. The track for the skip

is nailed to cross-ties *c*, which extend the width of the slope and rest against the upper side of the foot of the stulls. The stulls are tied together by a 2″ × 4″ stick, which also serves as a hand rail for the men, this being a two-compartment slope. The method of constructing the steps is shown in the upper right-hand corner of the figure, and consists in spiking brackets to stringers resting on the cross-ties. Boards are nailed to these brackets to form treads or steps.

Fig. 34

58. Pocket Shaft Stations.—In order to hoist from different levels of a mine, it is necessary to have loading stations with pockets for ore. A side elevation of an arrangement for this purpose is shown in Fig. 35. The pocket *a* is cut in the hanging wall so as to have an inclination of at least 40°. If the shaft is in the foot-wall, the pocket can take the direction shown to the vein; if, however, the shaft is in ore, the pocket must be nearly parallel with the shaft but not directly above it, as that would weaken the

roof above the shaft. In the figure, *a* is the chute, or pocket; while *b* is the loading lip or apron, which may be raised or lowered by the windlass *c* and the attached chain *s*. The platform *d* extends from the hanging wall to the foot-wall and below it is shown the skip *e* with its landing stop *f*. The lever *g*, which is worked from the platform *d*,

is attached by reach rods *h* to gates *m*, which in the position of repose form part of the slope track. Whenever it is desired to load a skip, the gates are raised by the lever and the wheels of the descending skip drop into a curved device *n*, Figs. 35 and 37. This device holds the skip under the ore-bin apron, takes the weight from the hoisting rope,

FIG. 36

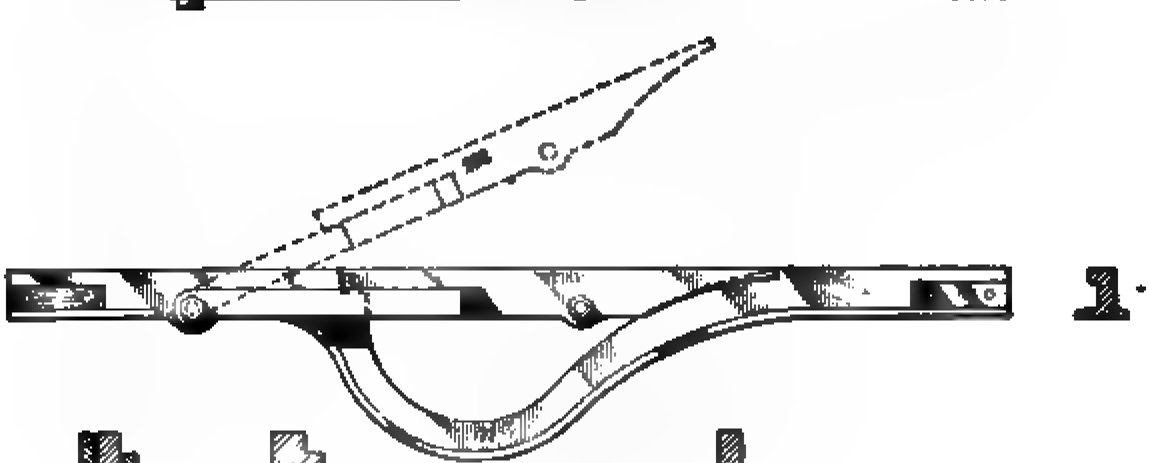


FIG. 37

and further tilts the skip so that it may be loaded to its capacity. The lever *o* is used to signal the engineer, when it is desired to stop the skip and hoist from this level. An elevation with the rock cut away on the line *A B*, Fig. 35, is shown in Fig. 36. The windlass *c* with chain *s* is connected

(a) Fig. 36

with a shaft *r* by sprocket wheels. The shaft in turning winds or unwinds the chain attached to the movable apron chute *b*. The figure shows a double-tracked slope, with the skip *e* being loaded on one track. Fig. 37 shows in detail the bridge *m* and stop-curve *n*.

59. Pocket Stations on Steep Slopes.—The advantages to be derived from loading stations are that the different levels in the mines can be worked at the same time and the output thereby increased, and that there need be no waiting for the trammers to come to the skip, and the trammers need not wait on the skip. This makes hoisting and loading independent somewhat of mining and tramming, and vice versa, for the pockets hold sufficient ore to supply the skip for a short time. Fig. 38 (*a*) shows one method of constructing a pocket, in which the timbers are carefully framed. The slope in this case has posts *a*; sills *b*, which also act as cross-ties; caps or collars *c*; and ties *d*. The space between these four-stick sets is lagged on the roof side. The uprights for the ore bin are independent of the slope timbers, although they can all be tied with bolts if it is so desired. The tramming for the bin and station must be carefully made, so that no lost motion will take place at the joints. Square-set framing, Fig. 38 (*b*), will be serviceable at all but the mud-sill joints *m*. Against these tenons the posts *a* abut by means of the tenons *c*, as well as the caps *b* by means of the tenons *d*. The ties are mortised in a manner similar to the caps *b*, and abut against *c* at right angles to *a* and *b*. The mud-sills are framed as in Fig. 38 (*c*), or in some equally serviceable manner.

60. Floor Pockets.—Pockets are sometimes made in the sides of the incline when that is steep. The system is shown in plan and elevation in Fig. 39. The ore pockets *c* are excavated in the foot-wall, if possible, on each side of the slope, so as to leave a solid wall beneath the track and do away with the timbering shown. The slope *a* is timbered with square sets and lagged on top. The ore is trammed on the levels *b* to the slope pockets and dumped. There is a hinged chute *e* connected with a gate in the pocket, by means of which the skid *d* is loaded. The pockets can be sunk in ore, and arranged so that little timbering is needed in their construction. Pockets are valuable adjuncts to slopes, since hoisting is made easier for the engineer,

who can pay attention to one set of pockets until they are empty and then hoist from some other level. This system

FIG. 30

of hoisting also minimizes the chances for accidents through the engineer mistaking the signals.

SLOPE SINKING

61. Factors in Slope Sinking.—Considerable difficulty may be experienced in overcoming the various troubles incidental to slope sinking unless every contingency is appreciated and provided for. Tracklaying also on slopes becomes more difficult as the inclination increases. Other factors than gravitation require that slope tracks be carefully laid and securely fastened. Drainage and the removal of blasted material require some form of power plant. The rate at which slope excavations may be advanced depends on the

power at command and the facilities for drilling holes, timbering, hoisting broken material, and pumping water. The necessity for sufficient power is further emphasized when it is understood that, before each blast, pumps and tools must be raised out of danger or protected in some way from flying rocks.

62. Slope Excavations.—While the character of the rock determines somewhat the ease with which it may be broken, yet in slope sinking the rock is difficult to break when compared with tunnel driving in similar rock. Nearly all blasting must be done against the action of gravitation, and additional drilling and powder will be required to break the rock if it binds. Some miners arrange the holes for slope sinking the same as for tunnel driving—something that is not always possible where much water is encountered. The usual method, therefore, is to keep the bottom of the slope in advance of the top, or else have one side kept in advance in order to collect the water. The center might be chosen for the sump cut; but as the latter is directly under the track, it is not advisable. The advantage to be derived from a side cut independent of the advantages accruing from the use of a sump is that two free faces are given for subsequent blasts, and in some cases lifting shots will be found serviceable. Advantage should also be taken of any slip or cleavage planes that make their appearance during sinking operations, in order that the breaking power of the explosive may be made more effective.

63. Placing Drill Holes.—In sinking slopes, the pitch will have some influence on the method followed in placing the drill holes. In flat slopes, as shown in Fig. 40 (*a*), it will be found better to work on a vertical face *a b* rather than on a horizontal floor *c d*, for the reason that, after the lower holes are blasted, the two free faces together with gravitation will assist the others. It would be found rather difficult to put in holes on the floor *c d*, as the roof would interfere with the drilling and the shots would in any case have a tendency to leave a more or less vertical face.

In steep slopes, a flat floor may be worked to better advantage at times than a vertical face. Fig. 40 (*b*) will permit the use of shaft columns in moderately wide veins or tripods for machine drills in wide veins. If a slope requires to be timbered in order to protect the men and machines, it

(b) (c)

Fig. 40

should follow the excavating closely. While timbering is being done, the track should be laid close to the broken material, and as soon as completed loading should be begun. In case it is necessary before blasting to remove the pump from its platform at *a*, it must be returned and possibly lowered to a new foundation. A good plan, particularly

when water comes from above the sinking operations, is to have a tank sump and collect the water, and then pump from this sump. The advantages of this arrangement are that the pump need not be disconnected before nor connected after each blast, and that the water does not run to the bottom of the slope and interfere with the loaders and drillers. In case water comes in below the tank sump, a pulsometer that is easily connected and disconnected can be used to raise water for some distance to the tank sump.

Fig. 40 (*c*) shows a vertical face in a highly inclined slope and the disadvantages drillers must work under. Tripods or shaft bars might be used in this case, but neither could be worked to advantage for every hole, and the tendency would be to eventually work to the flat floor, Fig. 40 (*b*).

64. Rope Mats.—It has been customary in sinking to weight down the shots with heavy timbers, and to cover the machinery with a battery of rough logs. The battery was usually placed a short distance up the shaft, and was a sort of platform to which the machinery and tools were hoisted. It was also the place to which the loaders and blasters retired when shots were to be put off.

A comparatively recent method for the prevention of rocks flying from a blast is that of placing a rope mat against the face and weighting it down. This mat system was practiced with great success during the construction of the subway in New York. Fig. 41 shows the mat construction. Hemp ropes are woven at right angles in such a manner as to leave small meshes. The ropes where

Fig. 41

they cross at *a* are fastened with stout twine, and the ends of the rope *b* are looped over a border rope *c*. The border rope is given a loop at *d*, in order to fasten the mat down and enable it to be hauled about. Ropes for this purpose should be at least 1 inch in diameter; sometimes, they are made larger than this, for the heavier they are the greater is the resistance they will offer. When the stones fly against the mat they are gradually brought to rest.

65. Power for Sinking Slopes.—Horse whims or horsepower hoisters are serviceable and useful machines where time is an object, but their usefulness is limited by the depth of the shafts or slopes and their size. While the hoisters named can raise heavy loads, they are geared back so that animals are not able to travel fast enough to accomplish the rapid hoisting needed.

When rock drills are used for sinking, steam power should be installed for the compressor, hoister, and pump. The boiler plant for the compressor could also run the hoister, since hoisting and drilling are usually carried on at separate times; but, even with no water to pump, timbers, tools, and tracks must be lowered, and these will require the use of some power over and above that necessary for the compressor. It is customary, therefore, to arrange the boiler plant so that the hoisting engine and pumps shall have separate boiler power from that of the compressor. Where large slopes are being sunk, several engines are employed—one for hoisting the broken rock; another for lowering timbers, pump pipes, tracks, and tools; and sometimes a third for the pump. The several engines are not of the same size or power, but are graded to meet the conditions of the work for which they are intended.

66. Tracklaying on Slopes.—Besides the regular action of gravity tending to pull the track down a slope, the thrust from the cars also acts in a similar manner. To these forces are to be added those due to changes in temperature, or the contraction due to cold and the expansion due to heat. The power exerted by the combined forces is sufficient at

times to break fish-plate joints; hence, when laying slope tracks too great care cannot be exercised in fastening them securely. In order that hoisting may be carried on while slopes are being sunk, the tracks should extend to the slope bottom. The lower 12 or 16 feet of the track can be made portable, so as to be quickly removed for blasting; the rest of the track, however, should be firmly fastened to ties and timbers.

Fig. 42 shows the method of laying and holding a track on a flat single-track slope. The ties *a* are placed between

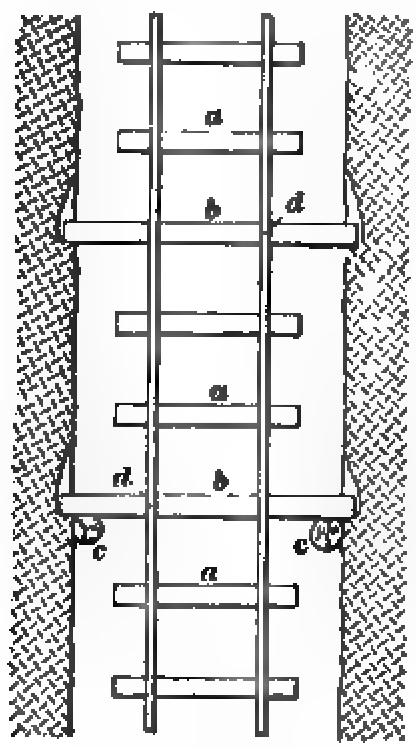


FIG. 42

FIG. 43

long ties *b*, which are held in place by hitches cut in the walls. In strong material, these hitches are sufficient; in weak material, however, they should be reinforced, as at *c*, by posts or by iron pins placed on the dip side. When laying tracks, care must be taken to prevent two rail joints from coming on the same cross-tie. It is considered almost imperative that the joints in mine tracks should come as shown at *d*, and even more so on inclines. The reason for this is that, if all strains came at one place, they would not have the same equalizing and stiffening effect that occurs when joints are broken, and hence rupture would occur much more easily

67. Double Tracks on Slopes.—Fig. 43 shows the plan of a double-tracked slope from 12 to 16 feet wide. If it is desired to divide the slope into compartments, posts *c* are placed between the tracks and the brattice boards *f* nailed to them. It will be noticed that every third tie is held by hitches cut in the side walls of the slope, and that the intermediate ties extend under both tracks. On steep stopes this reinforcement is not sufficient, and holes are drilled in the foot-wall into which iron or steel pins *d*, 1 inch in diameter, are securely fastened. There are two methods of securing these pins; namely, by wedging with wood or by calking with lead; the latter method while more expensive is probably the better. Any timber posts used on the slope are placed to the dip side of the ties, and should be inserted in hitches cut in the foot-wall to prevent them from being pushed out.

68. Compartment Slopes.—Where double-tracked slopes are used, they should have partitions between the tracks for the purpose of securing ventilation. This makes a two-compartment slope, but the same effect on ventilation can be brought about by putting the steam, water, and air pipes in a compartment separated from the hoisting. The heat radiating from the steam pipes will cause an upward current of air and thus increase the ventilation, the fresh air passing down the hoisting compartment. When a separate ladderway is required for the men, the air and pump column pipes can be placed in this compartment, and the steam pipe in the hoisting compartment. The steam pipe if placed in the traveling compartment would heat the men when ascending the ladder, and the upcast would give them impure air to breathe from the mine below. Both conditions are detrimental to health, for which reasons they are to be avoided.

69. Ladders and Man Engines.—If men are to travel up and down a slope, a compartment should be set aside for their use. The necessity for this increases with the inclination of the slope, since the men are more liable to accident

from any object falling down a steep slope, from the fact that they cannot so well get out of its way.

In some cases, the law demands that man engines be installed for the miners to use when going into and out of the mine. In other cases, the men are allowed to ride into and out of the mine on cages, cars, or buckets, ore hoisting being suspended during this time. That such laws are just, a short calculation will show. Take, for example, a slope with a thousand steps each 1 foot high, and say that a man 150 pounds in weight has to ascend them after a day's work. In doing so he will have performed 150,000 foot-pounds of work or an amount equivalent to raising 75 tons 1 foot.

70. Timbering Slopes.—The method of timbering adopted for inclines depends on the strength of the walls and the angle of inclination. Some slopes require very little timber; others need timbers their entire length. There are numerous ore deposits, the walls of which have been shattered by dynamic influences, particularly in mountainous regions, and a slope if excavated in such material must be timbered from top to bottom with timbers set close together. Some such formation as that just mentioned is usually encountered near dikes.

Mineral deposits are found in material of a clayey nature that swells in some cases and spalls in others. Some kinds of porphyry have been altered to clayey rock, which swells when an excavation is made in it. Some shales spall badly when air and water come in contact with them after an excavation is made. Water-bearing rocks, particularly limestone and sandstone, need to be carefully secured. Any swags in the roof show heavy ground, and should either be taken down or securely supported. About all the varying conditions met in tunnel driving will probably be encountered in slope sinking.

71. Skin-to-Skin Timbering.—Where ground is bad on moderately inclined slopes, it is customary to place timbers close together. Timbers for this purpose may be either round or squared, according to which form is the

most available and the most economical. The slope being a permanent opening should have durable timbers. If, instead of timber, masonry or concrete is used, it will be found in some cases to be an economical substitution. Fig. 44 is an example of three-stick round timbers placed skin to skin, the object being to resist the pressure of the mineral on the sides and of the hanging wall. This method of timbering is particularly serviceable at the entrance of a

FIG. 44

slope, as at this point the rock has little consistency, owing to weathering, which will increase after the slope has been opened.

72. Three-Stick Timbering With Lagging.—Where the pressure on slope timbers is not heavy, but still the rock is not sufficiently firm to keep from spalling, three-stick timbering with lagging is used. Fig. 45 shows the method adopted for a double-track slope that needs little timbering except for the minerals; on account, however, of its width —from 12 to 16 feet inside the timbers—it is considered unsafe to leave the top rock entirely unsupported. As the sticks for this method of timbering are from 10 to 12

FIG. 45

inches in diameter, it is customary to add a center post whose heel rests in a hitch cut in the rock and whose top is notched to fit the collar. The system of notching is a species of dovetailing that is made necessary on account of such sticks being

placed almost at right angles to the foot and hanging walls. The timber sets are given 36-inch centers, which space is decreased as the ground increases in heaviness. The lagging—whether sawed, split, or round—must be firmly fixed back of the timbers, particularly above the collars and joints.

73. Five-Stick Timbering.—When moderately pitching slopes are to be supported, a good job may be had if timbering such as that shown in Fig. 46 is adopted. This system is employed with entirely satisfactory results in some of the mines in the Eastern United States. The leg *a* is double-notched to the sill *c* and to the collar *b*. The post *d* has dovetail tenons at each end to mortise into the sill *c* and the collar *b*.

FIG. 46

Fig. 47 shows, in detail, the double notch for the leg *a* and the sill *c*. The latter is shown in the position it would have on the slope; that is, in the left-hand joint when one faces the slope at the surface. The leg *a* is reversed, to show the step in the notch that prevents the leg from slipping off the sill toward the dip.

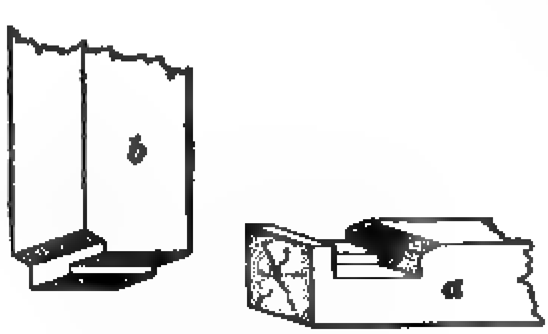


FIG. 47

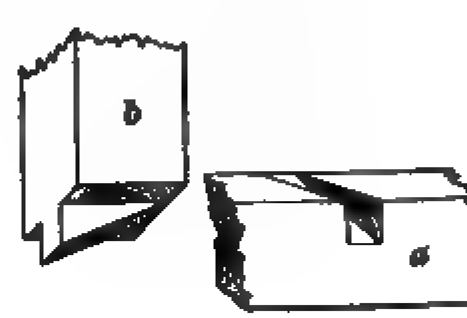


FIG. 48

Fig. 48 shows the wedge tenon for the post *d*, Fig. 46. The sill *a* is in the position it would occupy in the slope; that is, the deepest part of the mortise that becomes wedge-shaped is shown. The leg *b* is turned to show the tenon;

but the thickest part of the wedge fits into the deepest part of the mortise in the sill, thus preventing any movement of the post to the dip.

Fig. 49 shows the dovetail mortise and tenon that is employed a great deal for posts on slopes. This joint is readily made, and effectually prevents the posts from moving

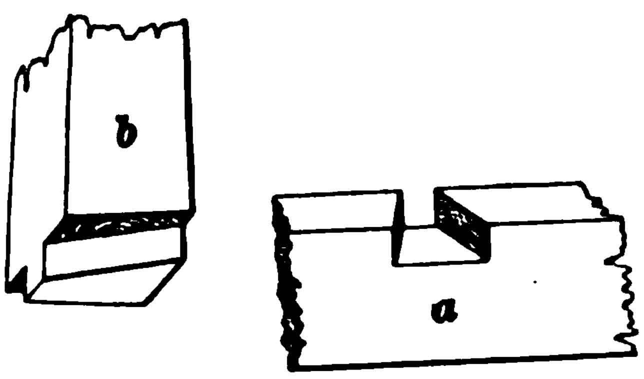


FIG. 49

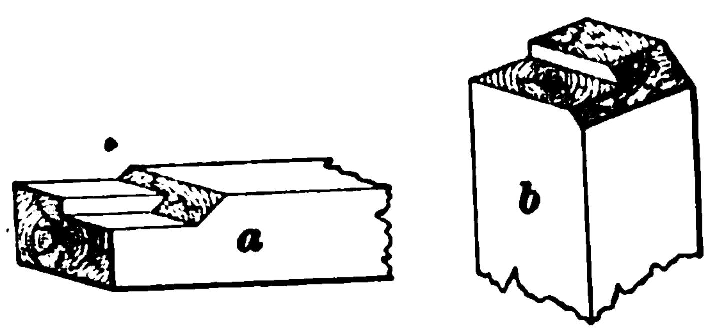


FIG. 50

down the slope. Care must be taken in making the cuts that the tenon on the leg will fit snugly in the mortise of the sill. If the tenon is too small, the pressure will not be transmitted to the sill properly, and this may cause the leg to be moved out of position. Joints that are loose must be keyed up rigidly by small wooden wedges. The same difficulties will occur in case the tenon is too large, but the remedies are different; for either the tenon must be cut down or the mortise enlarged. The joint between the collar *b* and the legs *a*, Fig. 46, is shown in Fig. 50. In the figure, *a* is the leg and *b* the collar turned over so as to illustrate the notching to the best advantage. It can be readily seen that this double notch is for the left leg of the timber frame when looking down the slope, and also that the raised projection on the leg *b* prevents the frame from moving unless the sill moves with it.

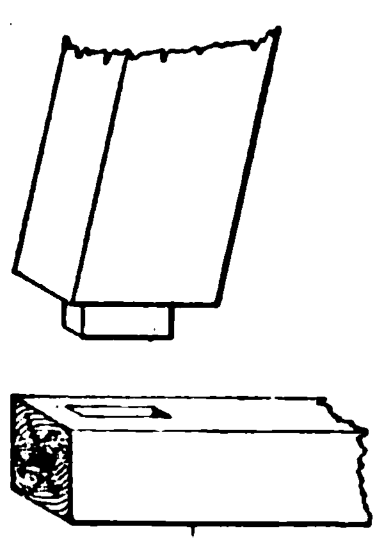


FIG. 51

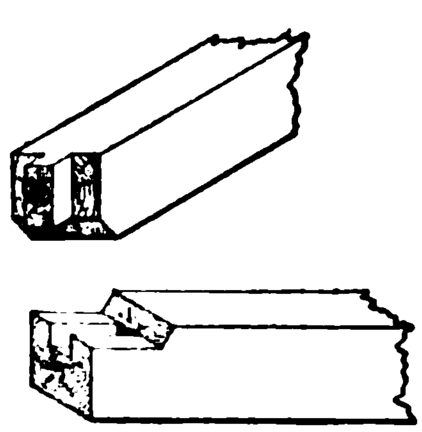


FIG. 52

Fig. 51 shows the ordinary mortise and tenon that is given bent legs and sometimes mine timbers; the notching

however, is faulty, since the pressure coming on the leg may cause it to move and shear the piece of wood between the mortise and the end of the stick. The joint shown in Fig. 52 is more serviceable than that shown in Fig. 51, besides being easier to make. The leg is kept from slipping out of the notch by wedges driven between it and the walls of the excavation, and such notching can only be employed where the timbers fit close to the side walls. If any other joint than those described is used on slopes, care should be taken that every part of it fits, for if poorly made, the weight will not come on it properly and may be thrown on a surface that will give way.

74. Timbering Steep Inclines.—Simple framed joints economize in material and labor, and it is therefore good practice to adopt a joint that will fulfil these requirements and meet the demands of the pressure. Steep inclines are timbered with square sets, the various members of which should be chosen with a view of accommodating the various pressures to be resisted. For example, Fig. 53 shows the side elevation of a square set for slopes. The posts *a*, sills *b*, and collars *c* are 10 in. × 10 in. square; while the ties *d* that separate the sets are 6 in. × 6 in. square. All the duty that the ties have to perform is to keep the square sets from moving out of place; hence, to use larger-sized sticks than are necessary to meet the requirements would be a waste of material. At an angle greater than 33°, the ties should be increased in sectional area, because they will have to sustain a part of the weight of the square sets. Above 32°, material will move by gravity down a rock slope that is fairly smooth.

FIG. 53

A plan of the timbering of which Fig. 53 is a side elevation is shown in Fig. 54. The caps *c* and the ties *d* are the only timbers seen. The ties may be dapped, or dovetailed, into the cap and sill pieces. In the former case, the dapping should also extend over the post ends, so that they will have the advantage of a brace, as shown in Fig. 55. The end elevation and plan show that the timbering is for a two-compartment slope. In Fig. 55 a center post 8 in. × 10 in. is shown. All three posts are dovetailed into the caps and sills. The dotted lines above and below the posts represent the positions occupied by the ties when in place. In case it is desired to make two compartments, a downcast and an upcast, the planks *e* are nailed to the central posts.

FIG. 54

75. Fig. 56 shows a system of timbering employed where the inclination of the slope is more than 45°. It will be noticed that the timbers are all the same size and nearly the same length. The posts and the caps *a* have cleats *d* nailed to them for the purpose of holding lagging *e*. In single-stick timbering on inclines, it is customary to give the posts or stulls an underset that is a slight inclination up the slope. Where square-set timbering is employed, the posts should be at right angles to the hanging and footwalls. If the joints are properly constructed, any movement of the hanging wall will be resolved into two components, so far as pressure due to weight is concerned, one

FIG. 55

FIG. 56

acting in the direction of the tie and the other in the direction of the leg; and these pressures will be transmitted to the foot-wall and other timbers. If there is an underset given to squared sets, the pressure cannot be transmitted so that the full cross-sectional area of the timber will be opposed to it.

The joint *A* in the side elevation is shown enlarged at *B*. The method of dovetailing the joints with posts, caps, and sills, and with ties, caps, and sills, is shown at *C*. From the top plan given, it will be seen that this timbering is for a three-compartment slope. The dovetailing of the posts is shown by the dotted lines, and the ties are shown dapped to fit into the caps and rest on the top of the posts.

PRELIMINARY OPERATIONS

(PART 2)

VERTICAL-SHAFT SINKING

SHAFT SINKING TO BED ROCK

1. Forms for Shafts.—In the United States, shafts at ore mines are generally made with rectangular cross-sectional areas. Shafts having such a form as this appeal to American miners, because they are more readily sunk, their walls can be cheaply supported by timbers, and every part of the space inside the shaft lining can be utilized. Shafts having cross-sectional areas in the form of rectangular parallelograms are preferred to those whose cross-sectional areas are square, as they can be more readily timbered, and the timbers are better able to resist side pressure at great depths on account of the ease with which they may be braced. This can be seen by referring to Figs. 1 and 2, which are rectangles having the

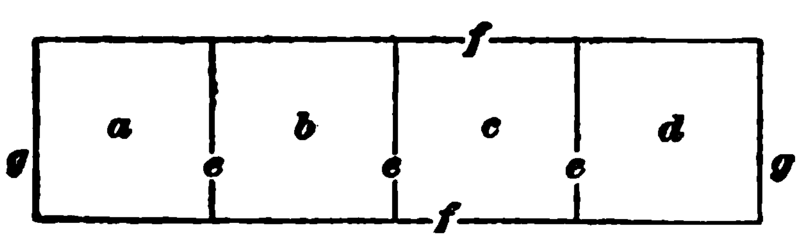


FIG. 1

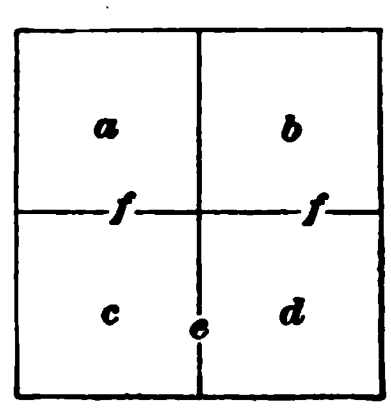


FIG. 2

same cross-sectional area. The compartments *a*, *b*, *c*, and *d*, Fig. 1, are in a row and are separated by timbers *g*, *e*, which act as braces to strengthen the wall plates *f*. The lining

that separates the compartments, also the cage guides in vertical shafts, and the anchors for pipes and ladders are fastened to the cross-pieces, or buntons.

The compartments *a*, *b*, *c*, and *d*, in Fig. 2, have the same area as those in Fig. 1, but the braces *e* and *f* are twice as long as in Fig. 1, and where bunton *e* is one stick, bunton *f* must be composed of two sticks jointed to *e*. Under such conditions, the buntons *e* and *f*, Fig. 2, would be as strong as those in Fig. 1, if it were possible to have a stick twice as long as another and just as stiff, or to join two sticks in such a way that they would be as strong as a single stick. The timbering is not the only objectionable feature to square shapes; in fact, the greatest objection lies in the width of the excavation, it being a well-established fact that the smaller a rock excavation is made, the less danger there is of its collapsing and the least trouble there is in securing it. The square and the rectangular parallelograms are not the theoretically correct forms for resisting pressure, nor will these forms answer for masonry where considerable side pressure is anticipated.

2. Circular Shafts.—In Great Britain and on the Continent, where openings are of a permanent character, masonry is used for shaft lining; for this reason round, elliptic, and polygonal cross-sectional areas are adopted. With the round and polygonal forms, there is considerable space in the shafts that cannot be utilized to advantage, except, perhaps, for ventilation purposes. Fig. 3 shows two cages *a* in a circular shaft, and the spaces *b* at the sides, only one of which, however, is made use of for pump column pipes *c*. In the elliptic or the nearly

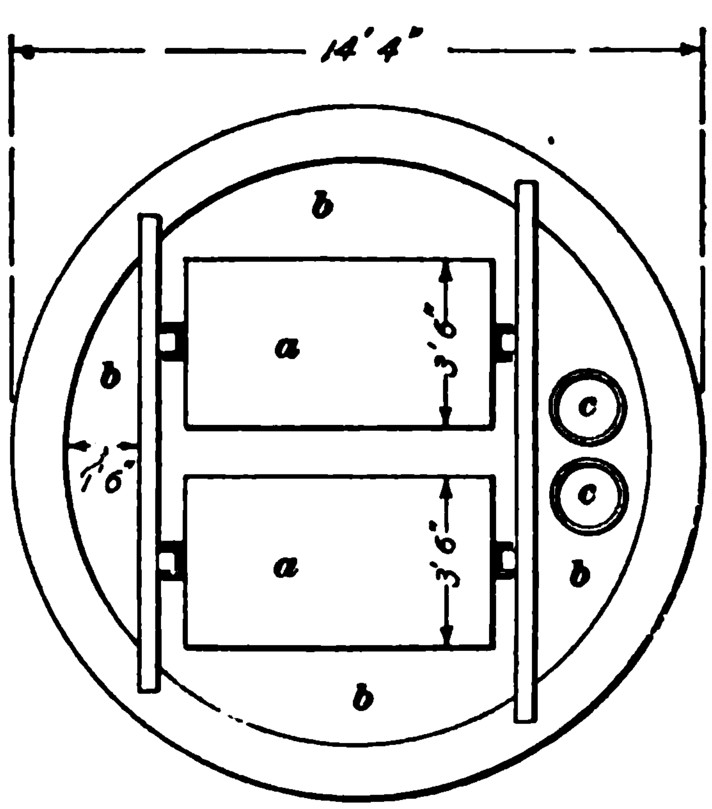


Fig. 3

elliptic forms, not so much space is lost as in the round or polygonal forms.

3. **Compartment Shafts.**—At ore mines having a small output, compartment shafts are not adopted because of the additional expense necessary for timbering. While this policy may be advisable during prospecting operations, no shaft mines should be worked without compartments, as they assist the ventilation. These remarks apply, even though only one hoisting cage is used. The advantages to be derived from better ventilation, and from having the pump pipes, air pipes, steam pipes, or electric wires separated from the hoisting, far outweigh the additional cost of timbering. Should anything go wrong with the pipes, or should it become necessary to examine and repair them, hoisting must cease during this time if there is but one compartment, and this may cause a shut-down at an inopportune time. An extra exit from the mine is provided by two compartments, and should a rope break or some other accident occur, the men may go into or leave the mine through the second compartment.

4. **Number of Shaft Compartments.**—The number of compartments in shafts depends on the size and the depth of the mine, and whether the intention is to hoist in balance. As stated, two compartments are advisable at any mine—one for hoisting and the other for a manway, a pumpway, or an upcast airway. Where hoisting is done in balance, three compartments are better than two, the third being used as a pumpway, a manway, or an upcast airway. Three-compartment shafts are almost a necessity in deep mines or in large mines where regular sinking and mining operations are carried on together, and where men and supplies must be taken up and down during working hours. In very deep mines, where hoisting is done from different levels, five compartments are sometimes used, as at the Tamarack mine, Michigan, and in some extreme cases six-compartment shafts are used. Usually two hoisting engines are worked at such shafts, and the output is quite large.

5. Shaft Timbers.—The principal timbers placed along the sides of shafts are termed **plates**; those on the longest sides of the rectangle, **wall plates**; and those at the ends of the rectangle, **end plates.** In order to divide shafts into two or more compartments, timbers known as **buntons** are placed horizontally across the shafts. Partition planks and cage guides are fastened to these timbers, the former usually by means of spikes and the latter by bolts whose heads are countersunk in the guides. Ladderways are not used in deep shafts, the men being lowered and raised in cages, or, in some cases, by special man engines. The latter are not often found in the United States, and at present probably there is but one, and that is at the Calumet and Hecla copper mine. In some states, the law requires that men shall be hoisted from the mines as well as lowered into them, and that supplies or tools shall not be lowered into the mines on cages with the men. The number of men that may ride on a cage at one time is also stated.

6. Size of Shafts.—A small mine will not need a large shaft, still a shaft having an area of 4 ft. × 6 ft. will cost as much to sink as one 5 ft. × 6 ft. in area. A shaft less than 6 ft. × 8 ft. over all is too small for rapid sinking, and should it become necessary to increase the size at some future time, the expense will be almost as great as sinking a new shaft. There should be a space of at least 6 inches between the cage and the shaft lining, to permit the air to rush past as the cage moves, otherwise the strains on the working parts will be increased when hoisting. Shafts vary in size from 4 ft. × 6 ft. to 15 ft. × 25 ft., the latter being exceptionally large. Table I gives the sizes of shafts at some well-known mines.

7. Shaft Allowances.—When a shaft is to be sunk, allowances must be made for the shaft lining or whatever timbering will be needed. In addition to the lining, buntons, partitions, and cage guides should be considered in the calculations. Nearly all wall plates are made of 10″ × 10″ timbers, while most buntons consist of 10″ × 6″ timbers,

TABLE I

SIZES OF WELL-KNOWN SHAFTS

Name and Location	Material Mined	Number of Compartments	Size of Hoisting Compartments	Size of Shaft Over All	Depth Feet
Centennial Eureka, Eureka, Utah	Gold, silver, copper, lead	3	4′ 2″ × 4′ 2″	5′ 6″ × 12′ 8″	1,610
Ontario, Park City, Utah	Silver	3	4′ 6″ × 5′ 0″	7′ 0″ × 20′ 0″	1,500
Red Jacket, Calumet, Mich.	Copper	6	6′ 3″ × 7′ 0″	25′ 0″ × 15′ 6″	4,900
Tamarack, Tamarack, Mich.	Copper	5	7′ 2″ × 5′ 2″	29′ 2″ × 8′ 10″	4,615
Anaconda, Butte, Mont.	Copper	3	4′ 6″ × 5′ 0″	20′ 4″ × 6′ 8″	
Butte and Boston, Butte, Mont.	Copper	3	4′ 0″ × 4′ 6″	18′ 4″ × 6′ 2″	
Hamilton, Iron Mountain, Mich.	Iron	6	4′ 8″ × 7′ 0″	23′ 4″ × 9′ 0″	1,460
Salisbury, Ishpeming, Mich.	Iron	3	5′ 6″ × 7′ 0″	22′ 0″ × 9′ 0″	1,400
Fayal Iron Company, Eveleth, Minn.	Iron	3	5′ 8″ × 5′ 0″	19′ 8″ × 6′ 8″	
Consolidated California and Virginia Mining Company, Virginia City, Nev.	Silver and gold	3	5′ 4″ × 4′ 6″	19′ 0″ × 7′ 8″	2,500
Virginius, Revenue Mountain, Colo.		2		8′ 0″ × 4′ 0″ +	1,400
Isabella, Cripple Creek, Colo.	Gold	3		13′ 6″ × 4′ 6″	940
Average for large mines in Colorado		3		15′ 0″ × 5′ 0″ 13′ 6″ × 5′ 0″	
Parker shaft, Franklin Furnace, N. J.	Zinc	3	5′ 0″ × 7′ 0″	19′ 1″ × 7′ 2″	980
General type, Joplin, Mo.	Zinc	1	5′ × 5′ to 6′ × 9′	5′ × 5′ to 6′ × 9′	
Richmond, No. 3, Scranton, Pa.	Anthracite	3	8′ 7″ × 12′	20′ × 38′	
Susquehanna, Hibbing, Minn.	Iron	3	5′ 6½″ × 6′	8′ × 18′	

unless the width of the shaft is more than 6 feet, when they should be as large in cross-section as the wall plates. The partitions are generally made of 2-inch planks, and the cage guides of 4″ × 4″ sticks, although 4″ × 6″ sticks are found in some shafts.

8. The size of a compartment should be calculated with regard to the factors named, and allowances should be made for spaces between the sides of the shaft and the cage. The size of the cage must conform to the size of the mine car and the number of cars the cage is to carry. The cage usually consists of one deck constructed for one car, but at some mines, in order to increase the output, it may be necessary to carry two cars side by side or end to end on a cage deck. This method is better than using double-decked cages because of the expensive landings required for the latter system, although in the main they are preferable to wide cages on account of the resistance of the air caused by wide cages moving in the shaft.

9. Locating the Shaft.—The mining engineer should locate the shaft in a position where it will reach the greatest area of the deposit. In some cases, the shaft can be excavated in the deposit; in other cases, it must be sunk in country rock and cross-levels driven from the shaft to the deposit. The Old Abe shaft at White Oaks, New Mexico, is 1,375 feet perpendicular, and at no time is it more than 30 feet away from the vein, which is almost perpendicular, sometimes leaning toward and sometimes away from the shaft.

In Fig. 4, which is a longitudinal elevation of a mine, the shaft is inclined at about 80° from the horizontal in order to follow the ore, and while the pitch is not absolutely regular, it is nearly so. The foot-wall is the better side for sinking when the vein is inclined, as in this position the inclination may be made uniform and the shaft can ordinarily be better supported than when in the deposit. The foot-wall offers another advantage; for instance, if the ore pinches out, sinking can be discontinued before vast sums of money are expended for work that will never bring returns.

When a vein is known to extend to a certain depth, as in the Lake Superior and Transvaal ore deposits, and a shaft

FIG. 4

is to be put down, it could be sunk in the hanging wall at such a distance from the outcrop as to meet the ore at the calculated depth. Few such cases, however, are known, until

the ore has been followed down by a previous shaft. Quite a number of deep shafts have been sunk to meet veins at a certain depth, but were not successful, as the deposit did not extend to that depth. The Geyser shaft near Rosita, Colorado, is probably the deepest exploration shaft in that state. It was sunk with the idea of tapping a vein known to exist on another property. The shaft reached ore, but not such as would pay to mine—at least mining is not being carried on.

If it is desired to remove the deposit quickly, two shafts are better than one, for the reasons that ventilation is improved and the miners and laborers can work faster, underground tramming can be accelerated, and supplies can be lowered to the mine workings without completely stopping hoisting operations. Such shafts should be placed at least 300 feet apart; but only after it has been proved that the ore extends that distance. Since a vein is likely to change in its dip or be faulted or peter out, the location of a vertical shaft should not be decided on until the deposit has been explored to an extent that will warrant the expense involved in sinking.

In order to obtain a uniform grade and at the same time explore the deposit, shaft *a*, Fig. 5, was sunk until it reached a vertical depth of 2,120 feet, at which point it encountered a disturbed area. The shaft *b* was next sunk vertically to a depth of 1,500 feet, where, on account of a fault and a change of dip in the deposit, it was sunk on an incline to a depth of 3,120 feet from the surface. It was presumed that shaft *c* could be sunk vertically; however, at 740 feet in depth a throw was met that necessitated an inclination, and again at 1,160 feet another throw, which required another change of inclination. This mine is in the Mysore gold fields, India, and is particularly notable owing to the fact that it is a deep mine—now 3,000 feet—that has never contained water. Sinking to the 4,000-foot level is in progress.

Work of exploration can be carried on in the deposit and probably pay its way; shaft sinking and cross-cutting, however, must usually be done in barren rock, which yields no

FIG. 5

returns. Mining operations should never be conducted in a haphazard way, but so that every dollar expended will net a return.

10. Commencement of Work.—The location of a shaft having been decided on, the ground should be graded by cutting away all inequalities on the surface; after this, the inside lines and measurements are to be taken by the aid of a transit and a tape measure. Next, twelve stakes are driven as shown in Fig. 6. The surveying instrument is set up at *1*, and stakes *1*, *2*, *3*, and *4* are driven in line, stakes *3* and *4* being placed at a convenient distance back from stakes *2* and *1*, so that they will not be injured during the operation of sinking. The distance between stakes *1* and *2* should be accurately determined, since they represent the inside measurements of

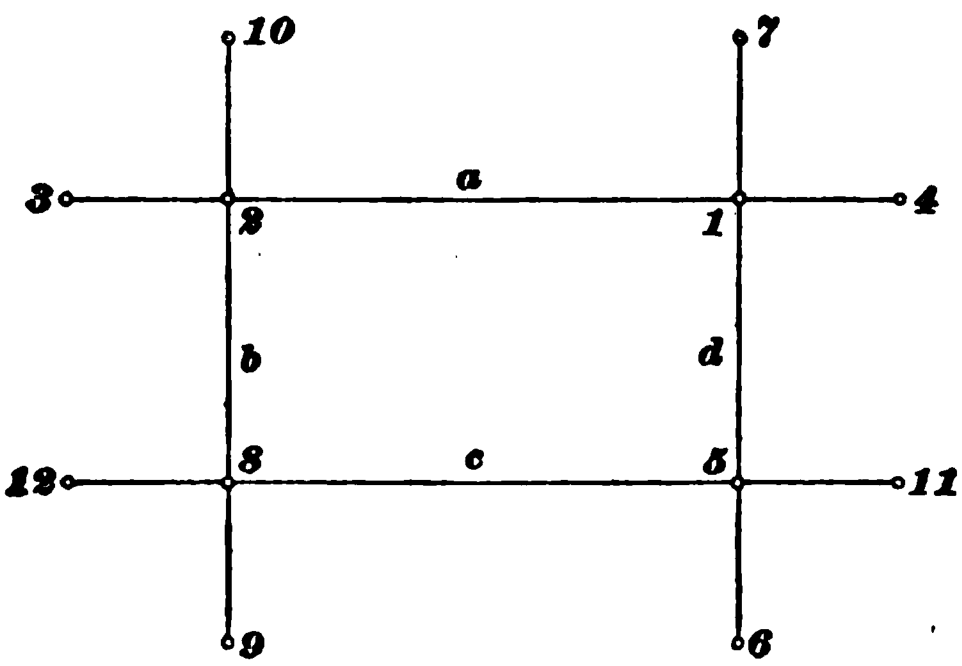


FIG. 6

one of the shaft sides. The transit is now turned 90° horizontally, and stakes *5*, *6*, and *7* are lined up and driven. The distance between *1* and *5* should be carefully measured, as it is the length of one of the shaft ends. The instrument is now moved to one of the other corner stakes, as *2*, and set up. A backsight is taken so as to bring *1* and *4* in the same line, after which the instrument is turned 90° horizontally and stakes *8*, *9*, and *10* are driven, the distance of the end *2–8* being carefully measured off equal to *1–5*. The transit is now placed at stake *8*, where a backsight is taken to stakes *10* and *2* and a foresight to stake *9*, to bring them in the same straight line. This being done, the instrument is turned

horizontally 90° and sighted to stake *5*, and stakes *11* and *12* are driven. The distance between *5* and *8* should agree with the distance between *1* and *2*; if it does not, there has been a mistake made. The tie-stakes *4*, *7*, *10*, *3*, *12*, *9*, *6*, and *11* will be found very convenient for reference marks as sinking progresses. Strings are now stretched between the outside tie-stakes, so as to bisect the center of the corner stakes, and digging is commenced. Trenches *a*, *b*, *c*, and *d* are excavated, and in them is placed the templet to line with the stakes.

11. Shaft Templets.—Shaft templets are carefully selected timbers so placed in trenches that they will line up with the cords stretched between the posts, as, for instance, *3* and *4*, *11* and *12*, *9* and *10*, and *6* and *7*, Fig. 7. These

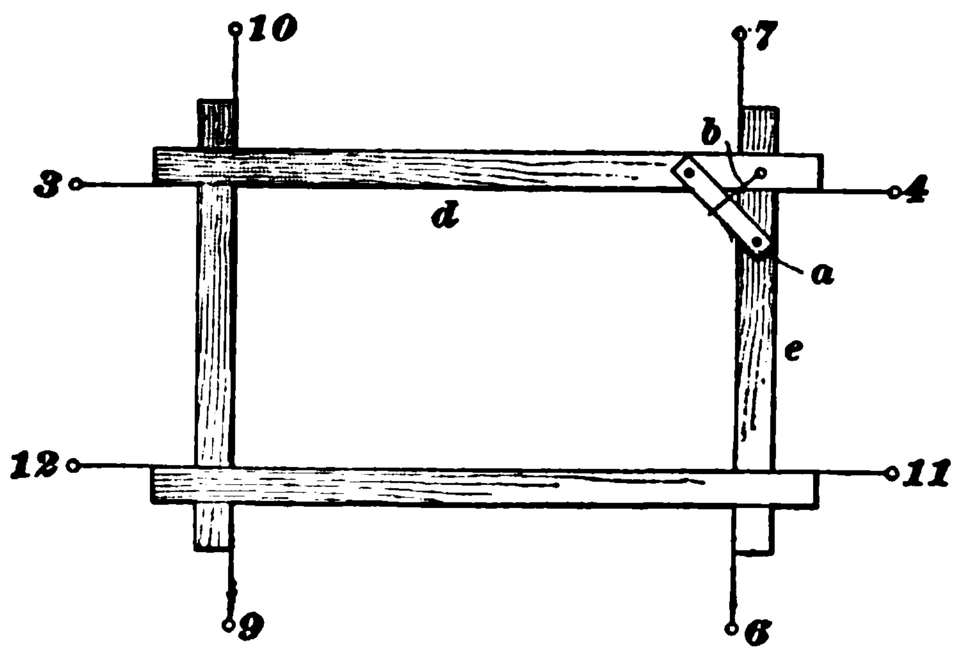


Fig. 7

timbers come in contact with the soil and will eventually decay; nevertheless, the end sills should be dapped with great care for 1 or 2 inches, in order to receive the side sills and hold them exactly true. When these timbers are in line with the strings, as shown, the inside measurement is the inside of the shaft. The timbers must now be pinned together and tamped about with clay to hold them stationary. The clay tamping acts as a roof, and causes surface water to flow away from the shaft opening. Care should be taken to prevent all surface water from running into the excavation, also to prevent all water hoisted from the shaft from seeping back in.

12. Sinking to Bed Rock.—The top soil inside of the templet is worked out with pick and shovel to a depth of 6 or 8 feet, which is as high as men can throw dirt readily. If the dirt is likely to cave, timbers are driven in position back of the templet, or are braced to keep it in place. A temporary head-frame is next erected, and a bucket, rope, and horse whim are installed. In large shafts, a boom and small steam hoister are preferable, since work will then progress much faster and the buckets can be swung and dumped on cars to carry the dirt out of the way. This work is continued until solid rock is reached, the timbering being carried down as the excavation proceeds. The excavation must be of sufficient size to receive the shaft timbers and to permit them to be lined with the templet. If the first timbers are simply temporary, allowance should be made in the size of the excavation for the permanent timbers that are to follow, which are placed inside of the temporary lining. For the purpose of making the shaft vertical, plumb-bobs are suspended from each corner of the shaft, as shown at *a*, Fig. 7. The plumb-line *b* is carried over a piece of ½-inch flat iron *c*, in which there is a nicely rounded notch to prevent the line from moving sidewise or from being cut by the iron. The flat piece of iron is securely bolted or spiked to the templet, while the plumb-bob line is fastened to a spike in the templet. Before the flat piece of iron is fastened to the templet timbers, however, measurements are made so that the line *b* will hang about 4 inches from the shaft plates *d* and *e*, and all timber-lining measurements are made from this line.

13. Sinking to Bed Rock in Wet Ground.—Sinking a shaft in a swamp or in the bed of an ancient river is a much more difficult undertaking than sinking in comparatively dry soil. Fig. 8 shows the elevation of the outer, or temporary, timbering for a shaft sunk in an ancient river bed. The sinking required heavy yellow-pine sticks 20 feet long for the end sills and two 20-foot sticks, spliced, for the side sills. The timber sets were suspended from two

trusses *a* that rested on concrete piers *b*. The bottom chords *c* of the trusses supported cross-pieces *d*, from which were suspended by iron rods *f*, the end plates *e*, while the side plates *g* were suspended from the chords *c* direct. After keeping the clay back near the surface by means of 2-inch planks, the shaft sets were put in place and separated by posts *h* 2 feet long placed at the corners and near the bolts. The latter were made of $\frac{3}{4}$-inch round iron, and were drawn up tight so as to form a rigid structure.

A system of forepoling was used for lagging, the forepoles *i* being made of $3'' \times 8''$ oak, 8 feet long, and pointed

FIG. 8

at one end. Wherever necessary, hay was driven with the lagging, and in this manner a comparatively water-tight cribbing was obtained. The contrivance for driving these oak planks was practically a slanting pile driver. It consisted of a channel trough greased on the inside, in which a short, heavy log was hoisted by means of a rope and winch located at the surface. When the log reached the desired height in the trough, it was tripped, thus permitting it to descend by gravity and drive the lagging down. With this device, it was possible to drive a ring of lagging in a day. The pressure on the sides of the excavation was sometimes enormous, and before reaching bed rock the cribbing was

forced in 12 inches, part of the 12″ × 12″ side pieces were cracked, and some of the posts were driven into these side pieces about 2 inches.

The removal of the material from the shaft was accomplished by the aid of a boom derrick, iron buckets, a steam hoist, and wire rope. At the surface, the buckets were discharged into dirt cars and hauled by mules to the dump bank. When completed, the shaft had three compartments —two 8 ft. 7 in. × 12 ft. for hoisting and one 12 ft. × 12 ft. for the airway. It is lined with concrete 3 feet thick, as

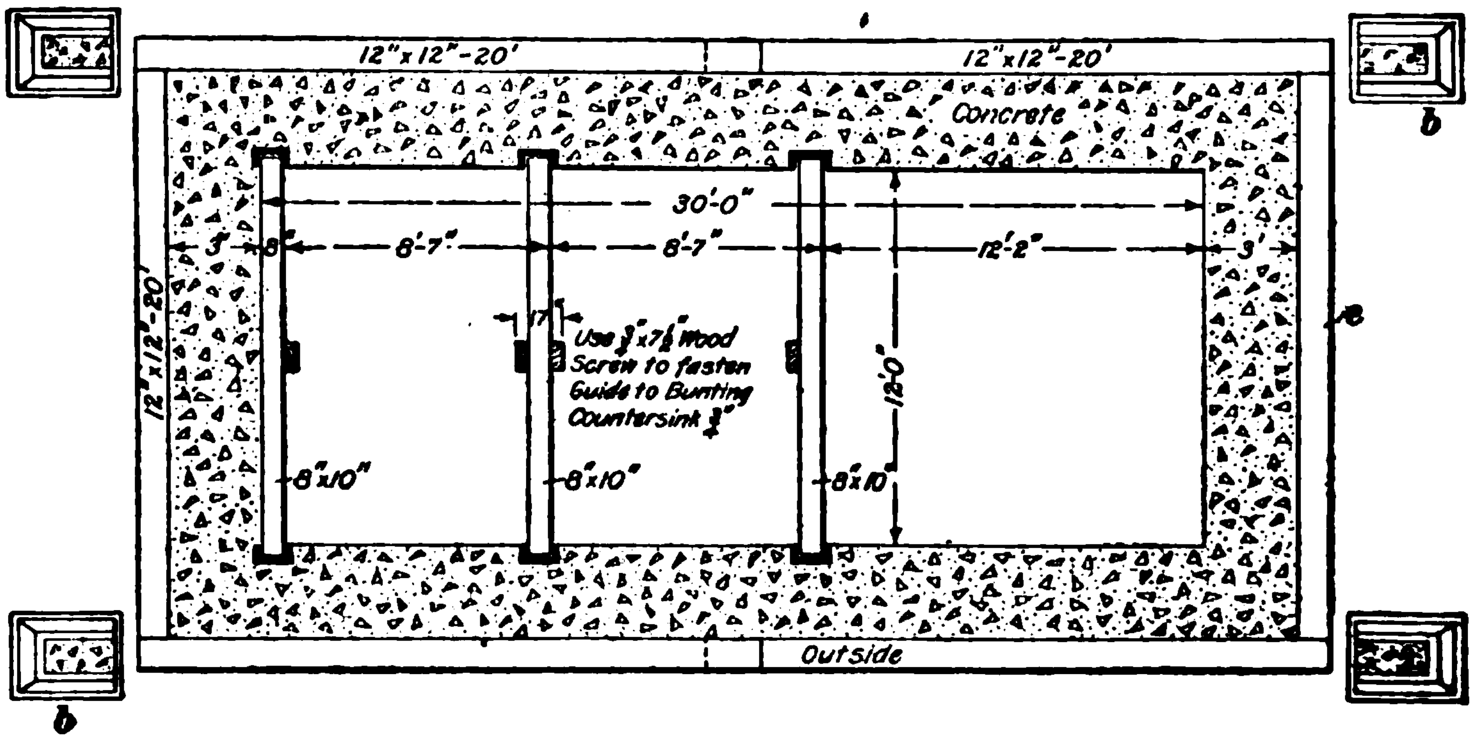


FIG. 9

shown in plan, Fig. 9, in which the letters have the same significance as those in Fig. 8.

14. Excavating Rock.—The preliminary arrangements to solid rock having been described, the next operation in order is that of breaking the rock and removing it from the shaft.

Wherever fuel and water can be readily obtained, it is customary to sink in rock with air drills, although in rare cases electric-power drills have been used. Hard, compact rock, such as granite, gneiss, sandstone, limestone, and some of the trap rocks, is easier to break than shales, slates, and seamy formations. The greatest expense attached to breaking ground is the cost of drilling, for which reason the sinkers must experiment to find the best positions to place

their shots. The cost of powder is also an important item, and its strength should be varied to suit the position of the holes and the rock to be broken. A seamy rock will not stand quick explosives, while a compact rock will; consequently, black powder will break more of the former rock than dynamite; and, on the other hand, dynamite will break more compact rock than black powder.

Holes fired in rock having but one free face require stronger explosives than those fired in rock having two free faces, and in shaft sinking, where air drills are used and holes are fired in rounds, this fact should be taken into consideration. Much will depend on the dip of the rocks in which the excavation is to be made, so that only a general rule can be given for placing the holes, and no definite strength of powder can be suggested, more than to state that in homogeneous rocks with one free face where an entering cut is being made, 60-per-cent. nitroglycerine powder is preferable, while for subsequent holes 40-per-cent. dynamite seems to produce the most economical results. In drilling, the holes should not be placed so far apart as to throw out rocks that must afterwards be sledged or blasted in order to load them into the buckets; for there is no economy in such work, in fact, it is more expensive.

15. Drilling by Hand.—When using hand drills for sinking in rock, the work cannot be carried on so rapidly nor so systematically as it can with power drills; the principle involved in placing the holes, however, is the same. It is customary, first, to make a cut in some portion of the floor, which serves the purpose of a sump and from which the water that accumulates is removed while the work of drilling, etc. is proceeding. This cut may be in the center line of the shaft or to one side, depending on the condition of the rock; but if there are no natural joints in the rock to favor side cuts, the entering cut is best made in the center or a little to one side, for then the blast can have the freest play. Men are greatly hampered in shaft drilling by water, the use of tools, and cramped room; and these features in connection

with hand drilling make shaft sinking slow work, particularly if the shaft must be timbered and the men are cramped for room.

16. Use of Air Drills.—The advantages derived from the use of air drills in shaft sinking are: speed in putting in holes; quickness with which the material may be reached after blasting, due to the use of compressed air in dissipating the powder smoke; the large quantity of material broken at one time and the consequent increase in advance over hand drills; and, finally, systematizing the sinking operations in such a way that the work moves smoothly. Quarry bars and tripods are generally used for the drills in wide shafts; but in narrower excavations, shaft bars up to 10 feet in length are employed. Steam power is not suitable for shaft sinking, and up to the present writing a first-class electric drill has not been devised. Much is said in favor of the exhaust from air drills as a means of ventilation; to some extent this is true, but, in shafts, the exhaust from the drills is charged with grease, and, being cool, remains at the bottom of the shaft and tends to keep the air impure. The exhalations from the men, and the carbon dioxide from lamps and possibly rocks are thus prevented from rising to the surface readily. The best ventilating apparatus for shafts is a blower.

Fig. 10

17. Arrangement of Drill Holes.—The positions of the drill holes should be such that a minimum number of holes will break all the ground necessary. It is customary to arrange these holes so as to break a center cut or side cut, and in wet shafts the cut holes are made deeper and are charged heavier than in other shafts, so as to leave a depression where water may accumulate for the pump suction pipe. Fig. 10 shows the center-cut holes *a* to be made longer than the other holes

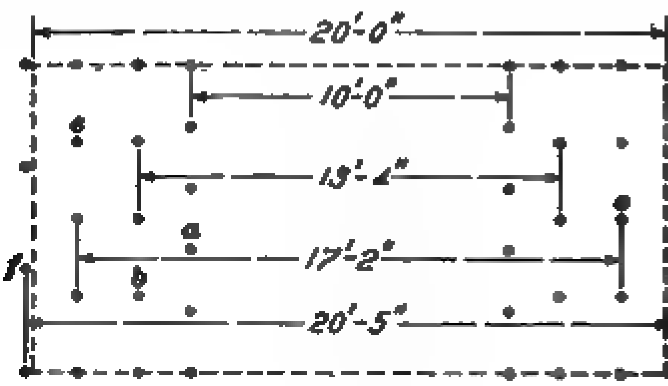
b and *c*, which will not break as deep when fired and thus leave a triangular cavity *e* for a sump. By making one set of the holes *c b* on either side of the shaft longer than the others and pointing them to a wedge, they will produce a cut, if fired first, that will answer as a sump.

18. The Parker Shaft.—The arrangement of the drill holes at the Parker zinc-mine shaft, Franklin Furnace, New Jersey, is shown in Figs. 11 and 12. The shaft was 20 ft. × 10 ft. outside, 16 ft. × 7 ft. inside the timbers. The drilling was done by four 3-inch air drills placed on two shaft bars, shown by the letters *c* and *d*. From the first set up, the center-cut holes *a*, six in number, on each side, were drilled at an angle that would furnish a depth of 6 feet perpendicular from the center line of the shaft. The drills were then set at an angle to put down the five holes *b* on each side of the shaft, after which the shaft bar was changed from position *c* to position *d*, and the holes *e* and *f* drilled. Considerable time is consumed in setting up tripods or shaft bars, but this system saved that time and permitted each drill to put down ten holes. After the drilling was completed, all the sinking tools, including the pumps, were removed and the center-cut holes *a* loaded and fired. If, after the broken rock was removed, it was found that the blast did not break to the bottom of the holes, what was left of them were charged and fired to break the remaining rock, in order to give the first round of side holes an opportunity to break.

In some cases, it may be necessary to blast the sump or center cut before the side holes are drilled. Such cases are unfortunate, since the machines, tools, and pumps must be hoisted, and then, after the blast, lowered again in order to resume drilling. This may sometimes be prevented by catching the water in the shaft above the drillers, in which event all the holes may be put in before any blasting is done.

The holes *b* and *e* were blown out with compressed air so as to remove water and small pieces of rock, after which the holes *b* were charged and fired, being followed immediately

by the holes *e*. After the broken rock from the previous blasts was removed, the side holes *f* were cleaned out with air, charged, and fired. In a tunnel, it would not be necessary to remove the broken rock before the following holes were fired; but, in sinking, the blasts are all made against gravitation, and the weight of the broken rock on the other rock opposes the action of the explosive. At the beginning, 6-foot cuts were made, but these were increased to 11-foot cuts by placing the holes as shown in Fig. 12. It was found,



Plan of 6 ft. Cut

Elevation of 6 ft. Cut

FIG. 11

however, that for cuts deeper than 8 feet an extra row of holes *a* inside the deep center-cut holes *b* was advisable.

19. Systematic Shaft Sinking.—While six successive cuts with holes placed as shown in Fig. 12 broke rock to a depth of 66 feet in the Parker shaft, Franklin Furnace, it would have been more systematic to have drilled all the holes and then loaded and fired them before any rock was

removed. Muckers could then have followed the blasting and remained at work until the rock was cleaned up, drillers could have continued at work until all the holes were put in the rock, and timbermen could have secured the shaft while the drillers were at work. This system could not be fol-

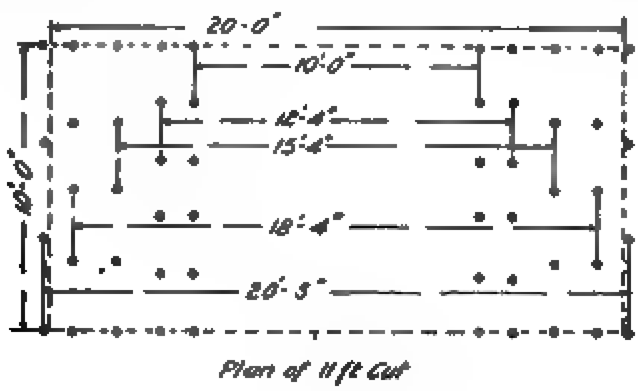


Fig. 12

lowed in the shaft named because of adverse conditions. With smaller shafts, the same system may be followed, but with fewer holes and drills. In this connection, it may be stated that such work as shaft sinking should be carried on systematically by two or three shifts, if necessary, and that the drilling should be continued until all the holes have the necessary depth. The loading and firing should then follow.

20. Channeling Cuts in Rock.—In some instances it has been found that if a few large holes are drilled 2 inches apart in a straight line and in the center of a cut having one free face, and the partitions between them are broken down by a suitable tool, considerable economy is effected, both in the cost of drilling and in the quantity of powder required. Fig. 13 shows the system used in Switzerland, which is said to have reduced the cost of sinking about $2.30 per foot. The holes *a* are first made and then cut to form a groove; then the center holes *b* are drilled, and afterwards the holes *c* and *d*. When holes *b* are fired, a space represented by the inside dotted line is broken. The holes *c* are then loaded and fired so as to break out that portion represented by the other dotted line; the holes *d* are fired last and give approximately the desired area of the shaft. With a modern rock channeler constructed to make a horizontal cut, it seems possible that this central groove could often be made without any more trouble than drilling the holes, but would depend, however, on the character of the rock.

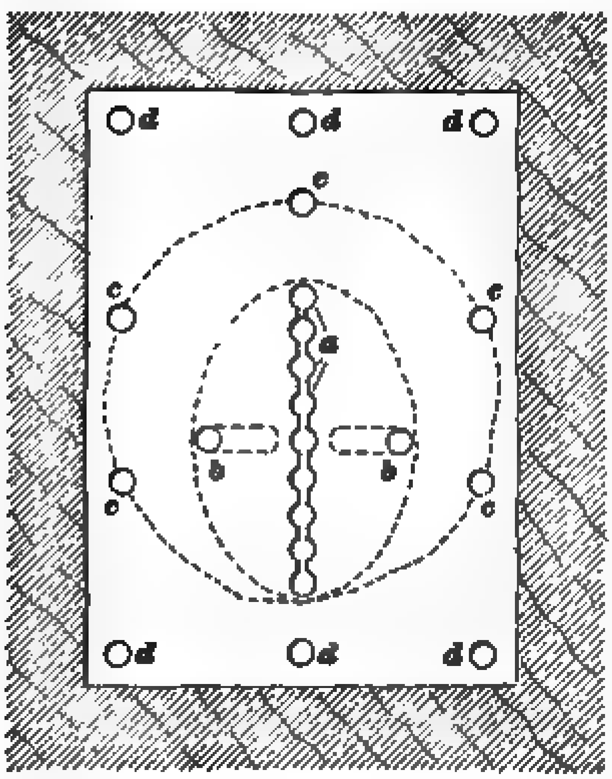


FIG. 13

21. Spacing Shaft Timbers.—Fig. 14 shows in elevation the timbering of a rectangular shaft having three compartments. Through the surface ground and down to solid rock the excavation is made larger than the area of the shaft in the rock, in order that it may be made water-tight and capable of sustaining the pressure from the wet ground. The timber sets for the soft ground are first put in and separated by short posts, termed *punch blocks*, after which they are

lagged by planks *c*. The permanent timbers are next put in place and planked outside by *h*, forming a box between

Fig. 14

the two sets of timbers in which masonry or concrete *d* is packed. When a shaft reaches hard rock, but little timbering is required, and if it were not for the cage guides and

the necessity of having buntons and stulls for carrying wate pipes, steam pipes, air pipes, etc., none would be used. I some cases, the timbering is reduced to stulls resting i hitches cut in solid rock, being placed in horizontal an vertical rows 4 feet apart, similar to the buntons shown in th illustration. Not much is saved by the latter method, a will be ascertained when the time and the expense of cuttin hitches and lining timbers are compared with the cost o timber sets in place.

22. Lodgments in Shafts.—In cases where water bearing rock *a* is encountered, it will be necessary to pu in a dam *k*, Fig. 14, of timber, and then fill the hole *l* bac of the timbers with well-rammed concrete. If the pressur is great, or the quantity of water is large, the dam is mad water-tight, and a reservoir is constructed back of it in th

FIG. 15

water-bearing rock. A pump station is made either below or above this reservoir, and the water is pumped to the surface. After the water-bearing strata has been passed

through, the timbers are spaced about 4 feet apart to the bottom of the shaft, provided of course no further bad rock is encountered.

23. Cribbing.—Shafts are sometimes lined with timbers laid one above the other so that they will form a structure termed a **crib.** The cribbing can be made of round or squared sticks arranged as shown in Fig. 15, being notched at the ends so that they cannot be pushed in by end or side pressure. When logs are used, the spaces between them will be so small that little backing is needed to prevent dirt or small pieces of rock from working out and falling down the shaft; when, however, squared sticks are employed, the spaces between them will be larger, and broken stone or lagging must be used as backing if the rock is loose or has a tendency to crumble. The lower timber *a*, which sustains the cribbing, rests at both ends on rock, and must be thoroughly blocked, as shown by the wedges *b*, to prevent it from sagging. The cribbing sticks *c* should be placed as close to the side walls as possible; and tamped with broken stone *d*, or, better, concrete.

24. Light Cribbing.—In localities where timber of a suitable size is scarce, the cribbing is sometimes built cheaper by using sawed lumber 1 in. × 12 in., 2 in. × 6 in., 3 in. × 6 in., or some other size. Framing is not necessary for sticks up to 4 in. × 6 in., since the ends can be spiked as the sticks are laid in position. Cribbing of this description should have a plank *A* spiked at each corner, as shown in Fig. 16, to prevent any inward movement of the sticks. Another plan is to spike these

FIG. 16

timbers at the ends, and then nail a board close against them. In some cases close cribbing is followed by 1″ × 10″ or 1″ × 12″ boards nailed together, particularly where there is water-bearing sand or gravel. The boards abut against one another alternately at the ends, as in Fig. 17, and being held together by spikes, make a strong shaft lining. This system is practiced to a considerable extent in Illinois—where there is wet sand that is almost equal to quicksand in its movements —with the exception that every other board is an inch or two wider than those between which it is placed. This arrangement furnishes an opportunity for the loose ground to rest on the wide board, as on a shelf, and thus gives a downward as well as an inward pressure. Inside the shaft, the boards are all lined up flush; but outside, where exposed to the ground, every other one is irregular.

Fig. 17

25. Stulls and Cribbing.—When the sides of a shaft need cribbing, the excavation is carried downwards as far as the miner thinks it is safe to go; then, as in Fig. 15, hitches are cut in the walls and the stulls *a* are put in position across the shaft. The number of these stulls depends on the number of compartments to the shaft, and no matter how many are employed, they must be placed level with one another and firmly wedged in the hitches. They must be in line with the shaft collar or templet, and adjusted by measurements taken from plumb-bob lines. Cribbing in log-cabin style is then built on them, extending up to the stulls that support the next upper section of cribbing or to the collar of the shaft. Any vacant space behind the cribbing is filled with small rock or lagging and dirt. The corners should be thoroughly wedged,

to prevent them from moving either in or out, and also to stiffen the timbers.

26. Framed Shaft-Timber Sets.—The usual method of securing shafts is by timber frames. Each frame consists of four pieces that are jointed at the ends. Between two adjacent frames *a*, Fig. 18 (*b*), are placed posts *e*, whose

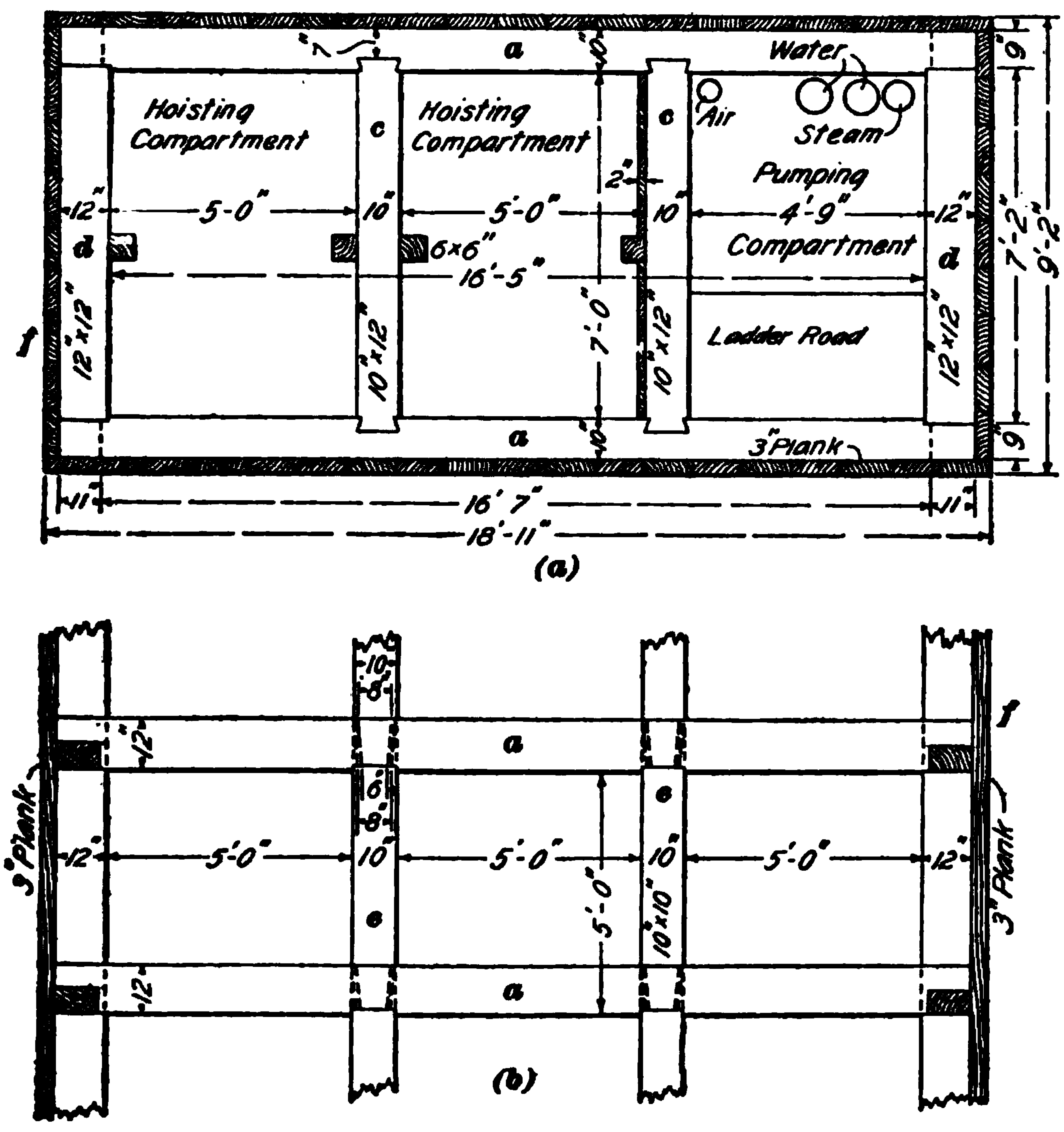


FIG. 18

length should not exceed 8 feet in any ground where the shaft is to be lined, and should decrease from this to 6 inches, according to the character of the ground. If the ground is very weak, the frames are put in close together without the use of corner posts, and act as frames and lining. For ordinary pressures, the joints are made as in Fig. 19, but for heavy pressure, the end joint shown in Fig. 20 (*b*) is

preferable to that shown in Fig. 19, although the buntons are tenoned alike in both cases. Fig. 20 (*a*) shows one of the timbers with its edge beveled, so as to form a joint that is strong and not so liable to split as where the notches are square cut. The complete joint is shown in Fig. 20 (*b*), and is equally capable of resisting pressure whether it comes from the ends or the sides, or both.

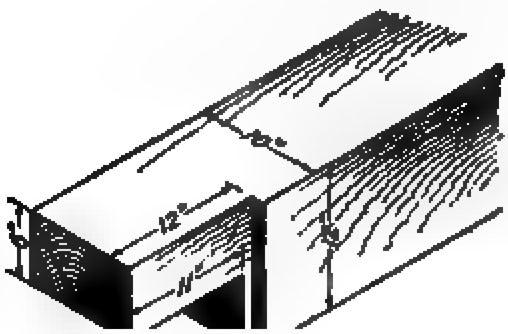


FIG. 19

Fig. 18 (*a*) is the plan of a three-compartment shaft at an ore mine. The side plates *a* are made of 10″ × 12″ timbers, the end plates *d* of 12″ × 12″ timbers, and the buntons *c* of 10″ × 12″ timbers dovetailed into the wall plates, as shown by the notch on stick *c*, Fig. 19. The lagging *f* consists of 3-inch planks, not because it is necessary to sustain pressure, but to prevent any loose stones from falling down the shaft. It will be noticed that the posts *e*, Fig. 18 (*b*), consist of 10″ × 10″ timbers 5 feet long. The combined ladder road, airway, and pumping compartment is partitioned off from the hoisting compartments, and thus acts as an upcast airway.

27. Advantages of Framed Timber Sets.—Square-set shaft timbering has certain advantages not possessed by cribbing. For instance, there is less difficulty in plumbing the shaft; in fitting the cage guides; in repairing and replacing timbers; and, if forepoling is necessary during sinking, it may be done with less trouble. Another advantage

that framed sets possess over cribbing is that they may be put in from the bottom up or from the top down, while cribbing must be built upwards from below. Fig. 21 is a perspective of three sets of square timbers with posts and buntons. The end plates *B* in this case are given dovetail tenons like the buntons *C*, in order to resist side pressure, no end pressure being anticipated. The side plates *A* are also dapped for the posts *D*, so that the latter may resist side pressure. There are many styles of joints used to resist the pressures that come from various sides, and, as they are described in another Section, will not be repeated here. The

(a)

FIG. 20

FIG. 21

timbers for square sets vary from 12 in. × 12 in. to 8 in. × 8 in The buntons and posts are usually somewhat lighter and

smaller than the wall plates, but even the latter may vary at the ends and sides.

A good form of joint for wall plates, to resist pressure coming from the side and end, is shown in Fig. 22. The bevel D in the wall plates A and B necessitates a small tenon on post F, and this will prevent the removal of the wall plates; whereas, the posts in the square sets, Fig. 21, can be removed readily and permit the wall plates to be removed as well. The object of the strip S nailed along the center and back of the plates is to form a shelf for the pieces of plank lagging to rest on. Even with this rest the packing between the lagging and the lining must be thoroughly done.

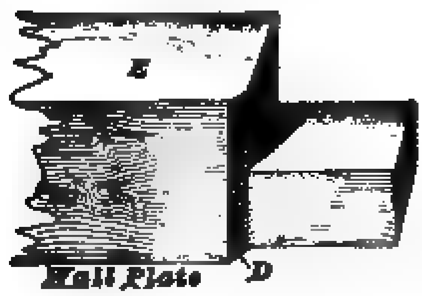


FIG. 22

28. Framing Buntons and Posts.—In Fig. 23, A is a post having a shoulder cut on it so as to fit the wall plate where tenons extend only part way across. The object of the shoulder is to form a support for the bunton under some conditions; for example, if the bunton is tenoned to go into the plate mortise from below. When the timbering follows close to the sinking, the buntons are likely to be in the way of the men, but when tenoned to go in from below, they may be fitted in the mortises after the side plates for another set are in place. Considerable chiseling is necessary at the corner B of the bunton to form a close-fitting joint of this description, but the tenon may be much simplified by sawing off those corners and leaving the tenon in the shape of a keystone, or wedge. In the latter case, it is possible to

remove any of the buntons without removing the posts, provided they are set in with the small end of the wedge down.

29. The Forman Shaft.—The shaft timbering on the Comstock lode, in Nevada, was exceptionally good, and several systems of jointing were introduced, some of which were never greatly improved. One L-shaped shaft, known as the Forman shaft, Fig. 24, was sunk for the purpose of separating the hoisting from the pump and other compartment. With the exception of the strips on the back of the side plates, the method of timbering is explained by the illustration. These strips are made of 2″ × 2″ or 3″ × 3″

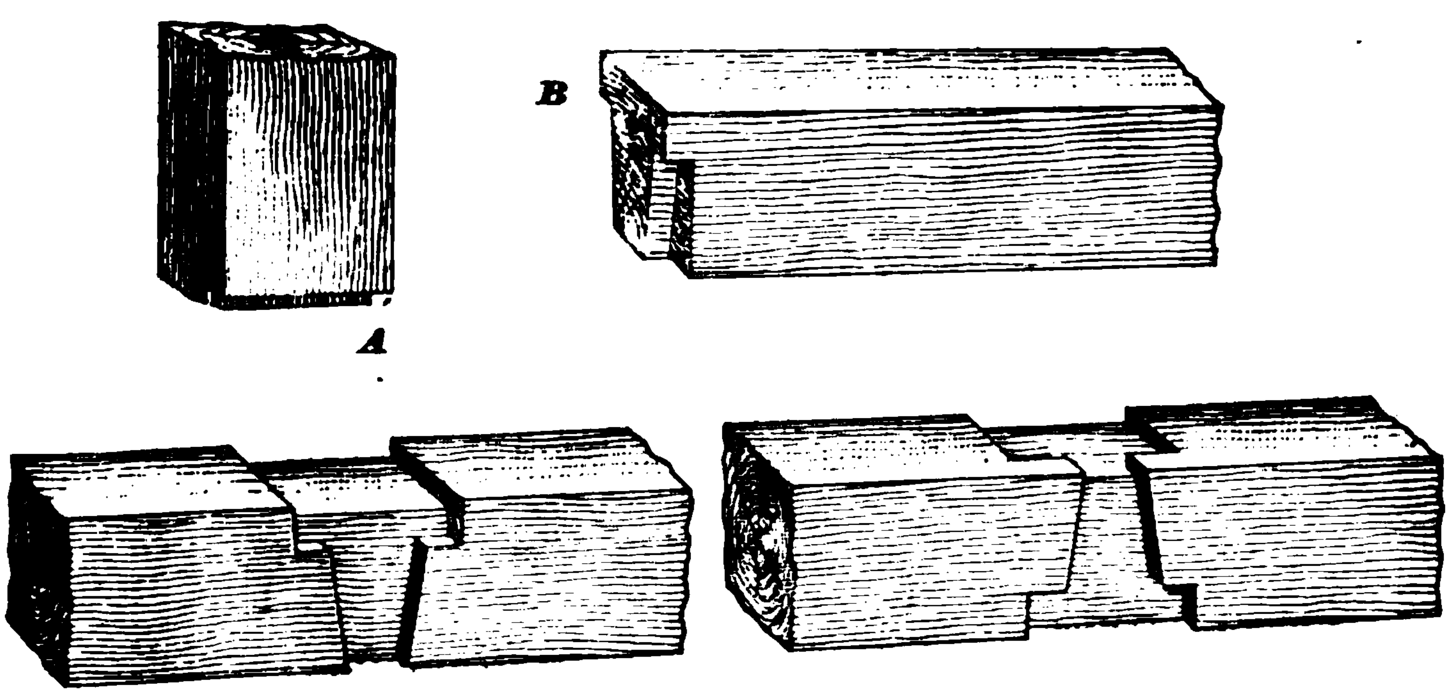


FIG. 23

lumber, being spiked as shown to hold the lagging that rests on them. The lagging consists of sawed material, which is made in proper lengths so as to fit in between the strips on the adjacent plates. As will be seen, the end plates are similarly supplied with strips for lagging that is placed outside the frame. In the figure, *a* are the posts; *b*, the wall plate; *c*, the long end plate; *d*, the short end plate; and *e*, the detail of the corner joints. Probably this is the only L-shaped shaft in existence.

30. Hanging Shaft Frames.—When the timbering is done from the top downwards, the timber sets are suspended by means of hanger bolts having a bent hook on one end and a thread for a nut on the other, as shown in Fig. 25. Two hanger bolts are needed to make a hanger, and each

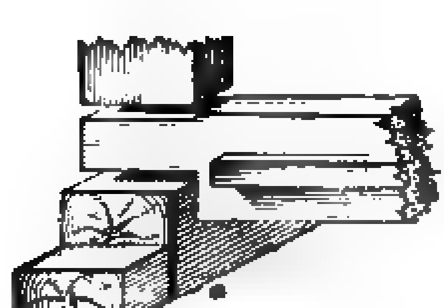


Fig. 26

compartment is fitted with one hanger on each wall plate, while two hangers are supplied for each end plate. For ordinary work, the hangers are made of 1-inch round iron, but for use in wet ground they are made larger or reinforced. The first set of timbers may be suspended from the shaft collar, or templet, or they can be hung from the first set that rests on solid rock below the templet, this position depending on the material at the shaft mouth. The hanger bolts are inserted from below into holes in the frame above, and after slipping a heavy cast-iron washer over the end of

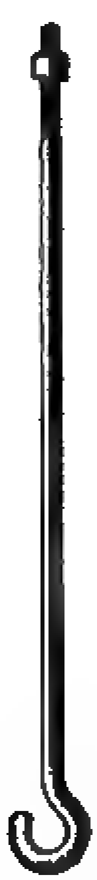


Fig. 25

Fig. 26

each bolt, the nuts are screwed on. The wall and end plates for the next set are then lowered, the hanger bolts with the hook ends upwards being inserted through the auger holes from the top. The two hooks are brought together and the washer placed on the bolt, after which the nut is drawn up loosely. The side and end plates are adjusted to their proper positions by the aid of plumb-lines and blocking,

and the posts are then put in place and the nuts screwed on the hangers sufficiently tight to hold them until they are again lined up. After the timbers are properly adjusted and wedged at the corners, the nuts on the hangers are screwed up tight. Fig. 26 illustrates a number of wall plates and their posts as they appear when hanging down a shaft.

SHAFT SINKING IN LOOSE GROUND

31. Cribbing for Loose Ground.—A form of support for excavations that will not stand alone or have a tendency to cave shortly after sinking is shown in Fig. 27. The end

Fig. 27

and side plates are placed skin to skin, and the buntons *B* are put in close together to prevent the wall plate *A* from

Fig. 28

bulging inwards. The buntons are arranged to break joints with the wall plates, and are placed in mortises cut into the plates about 1 inch, as shown. If the wall plates are long,

they may be in two sections, with a scarfed joint that comes behind one of the buntons, but each stick should not be of the same length—that is, should break joints behind the same bunton. Where only one bunton is used, however, every other stick must be the full length of the shaft, even if it is difficult to handle on account of its length and weight.

32. Plank Cribs.—In small shafts where buckets are employed or where the shaft is a temporary makeshift, boxes constructed of planks similar to those shown in Fig. 28 may be found to meet requirements. The planks are made of 2- or 3-inch stuff and are placed edgewise, being secured by blocking behind them, and also arranged so as to break joints. If strips of 2″ × 4″ stuff are nailed to the inside corners parallel with the length of the shaft, they will prevent the sides from being pushed inwards, and no other fastening will be required. The illustration shows the planks notched at the corners, but this is unnecessary where strips are used.

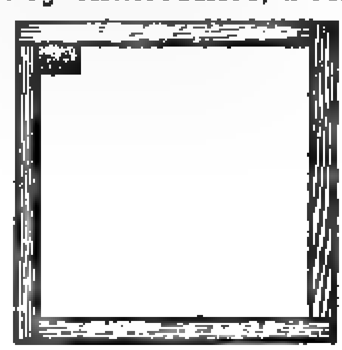


Fig. 29

Another system of lining for a small shaft or riser, which may be adopted where buckets are used for hoisting, is shown in Fig. 29. While this lining and the one shown in Fig. 28 are primarily for temporary use when prospecting or exploring, they may be adopted for winzes or raises in crumbling ore. The lining illustrated in Fig. 29 is jointed, and the planks are so cut as to make the box stiff by having one plank act as a brace for the other. In this case, unless the side pressure is very heavy, it will not be necessary to

reinforce the corners with strips placed as shown at the corner of the plan. The shaft illustrated in Fig. 29 measures 3 feet 6 inches outside and the shaft lining rests on the timber sets placed at the bottom of the shaft in the level. The material sunk through where this timbering is in use is clay that covered zinc deposits; similar shaft timbering is used in other situations, however, although the dimensions are different. The box in the corner of the plan is for the purpose of separating wires and air or steam pipes that are needed for conducting power below ground, and when so boxed, they will not be injured by the bucket.

33. Sinking in Loose, Wet Ground.—When the ground to be excavated is wet as well as loose, the problem of supporting the excavation becomes more difficult to solve as the quantity of water increases. There are many degrees of wetness, and as the pressure increases with the depth, the most troublesome excavation will be in water-saturated, loose ground, such as quicksand, which demands special methods for excavation, as well as for supporting it.

34. Forepoling.—When the ground is wet and liable to cave, but still not wet enough to run, the method of supporting the excavation known as **forepoling** can be adopted until solid ground is reached. Such timbering as that shown in Fig. 30 would not be suitable for more than a temporary makeshift, particularly if the ground passed through had more than a moderate depth of 25 or 30 feet, for, it would at times no doubt become saturated with water and exert a pressure on the timbers that they could not resist. As shown in the figure, the timbering is started from the surface, the first set *A* being put in position and the lagging *D* driven about it with a mall. The ground is then excavated inside of the timbers until the depth decided on for a second set *J* is reached, when that is put in and suspended from the set above by the planks *F*. In such cases, there should be mud-sills *H* to support the timber bearers *G*, and both the first and the second timber sets should be suspended from the latter, otherwise any cave would throw the timbering out of

line. Bridges C are sometimes fastened to the lagging D, and again are only placed between the forepoles K and D, their object being to prevent D from coming in contact with K and thus furnishing a better angle for driving the poles K. After poles K have been driven, the excavation is

FIG. 30

continued until the proper distance for another shaft set is reached, when it is hung from the set J in the same manner that J was hung from A, and forepoling continued for the next set. The shaft sets are spaced and kept in their proper positions by the corner posts B.

35. Double-Lined Shafts.—Since the shaft just described would have water trickling through the joints in the spiling and running down the shaft, it is customary to replace such temporary timbering by that shown in Fig. 31,

particularly if the shaft is to be large and permanent. As shown in the figure, this shaft consists of three compartments, which are separated by the buntons *B*. The wall plates *R* are placed close together, and between them and the lagging *C* is left a space that is packed with puddled clay or cement grout to make the lining water-tight. This system has proved satisfactory near the surface, but would not be effective for depths much below 100 feet. Where con-

Fig. 31

siderable pressure is to be resisted, concrete made of hydraulic cement and tied with iron rods is used instead of puddled clay.

36. Piling Through Soft Ground.—If the ground near the surface is composed of running material that will not stand alone, and comes up through the bottom of the excavation, the depth of the ground must be ascertained before work is commenced. Piles are then driven at a distance away from the shaft collar that will admit of other piles being driven inside of them, and thus preserve the proper size for the shaft until solid rock is reached. Fig. 32

illustrates the position of the piles *a* with reference to the shaft lining *b*. All the earth between these piles must be excavated and the piles braced laterally; this is not shown fully in the illustration, nevertheless the strength of the bracing must increase with the depth. The excavation becomes smaller with the depth, and thus the conditions for opposing the increased pressure are naturally obtained. This system of dealing with a difficult problem may prove satis-

Fig. 32

factory under some conditions, particularly where stiff iron girders can be used for braces until the shaft lining is in place.

37. Breast Boards in Shafts.—In some cases, so as to keep the material from entering the excavation from below, the bottom of the shaft is covered over with short planks that are held down by posts of different lengths. Fig. 33 shows a system of spiling in ground that must be held back by breast boards *a*. The work is slow, and

requires that the ground be excavated in sections, for which purpose the spiling *b* is first driven, the ground being excavated a little at a time until the desired depth is reached. This is accomplished by scooping out a small quantity of earth from underneath a plank and then blocking the plank in its new position. Sometimes, the ground is so fluid that the planks cannot be lifted, in which case they are advanced by jacks, and the material pushed back or allowed to ooze up through holes in the planks made for that purpose. The

FIG. 33

compartment in the center of the excavation is always kept ahead of the other work and takes the place of a sump. In some instances, where the water is pumped continuously from the sump for a day or more, the excavation can be carried forwards with less difficulty, as the pumping seems to drain the material.

The next operation is to widen out the center compartment, and for that purpose the spiling *c* is driven; then, the breast boards *a* are advanced a little at a time and blocked as shown at *d*. This operation having been carried on until the depth

desired depth is reached, the spiling *e* is driven down, and the excavation is widened out between the spiling *c* and *e* by advancing and blocking the breast boards in section *f*. The sections *g* and *h* are then widened, and proper timbers, such as the stringer *i*, are placed across the excavation and blocked. The center compartment *j* is then carried down as explained, and the shaft timbers *k*, *m*, and *n* are placed.

If the ground can be drained and there are no running streams, the operation of sinking through quicksand by spiling and the use of breast boards may be accomplished successfully in some cases; but the modern methods devised, although more expensive on account of material, are cheaper in the end, owing to the saving in labor and time; besides, when finished, the results are more satisfactory and in most cases the work is probably more secure.

SINKING SHAFT LININGS

38. Sinking Shafts by the Rothwell Method.—The Rothwell method of penetrating quicksand was devised for the purpose of reaching the sulphur deposits near Lake Charles, Louisiana. These deposits are covered with water-saturated silt that cannot be driven through by ordinary means. The system was not adopted; nevertheless it has some commendable features. The device consists of a hollow, pointed shoe *a*, Fig. 34, bolted at *b* to a hollow casing *c*. The joint *b* is made water-tight, so that water may be pumped from the surface into the shoe and spurted into the ground about the shoe. By means of the water jets, the casing can be made to sink for a distance by gravity, but when a boulder or hard pan is reached, jacks *d* are used to force the shoe down. As the shoe sinks, sections of metal lining *e*, *f* are added at the surface, thus making a hollow compartment *g*, down which men may go to work the jacks. No material is excavated, so that the pressure on the inside of the lining is as great as that on the outside. If any excavation is attempted, even to pumping out the water, the sand will pack about such linings and prevent any advancement,

Should the shoes strike an obstruction, the shaft lining will move out of plumb if the obstruction is not removed; hence, the jacks must be used to keep the lining plumb, and by the aid of the water jets force the obstruction out of the way. These jacks obtain a purchase against the shoe by

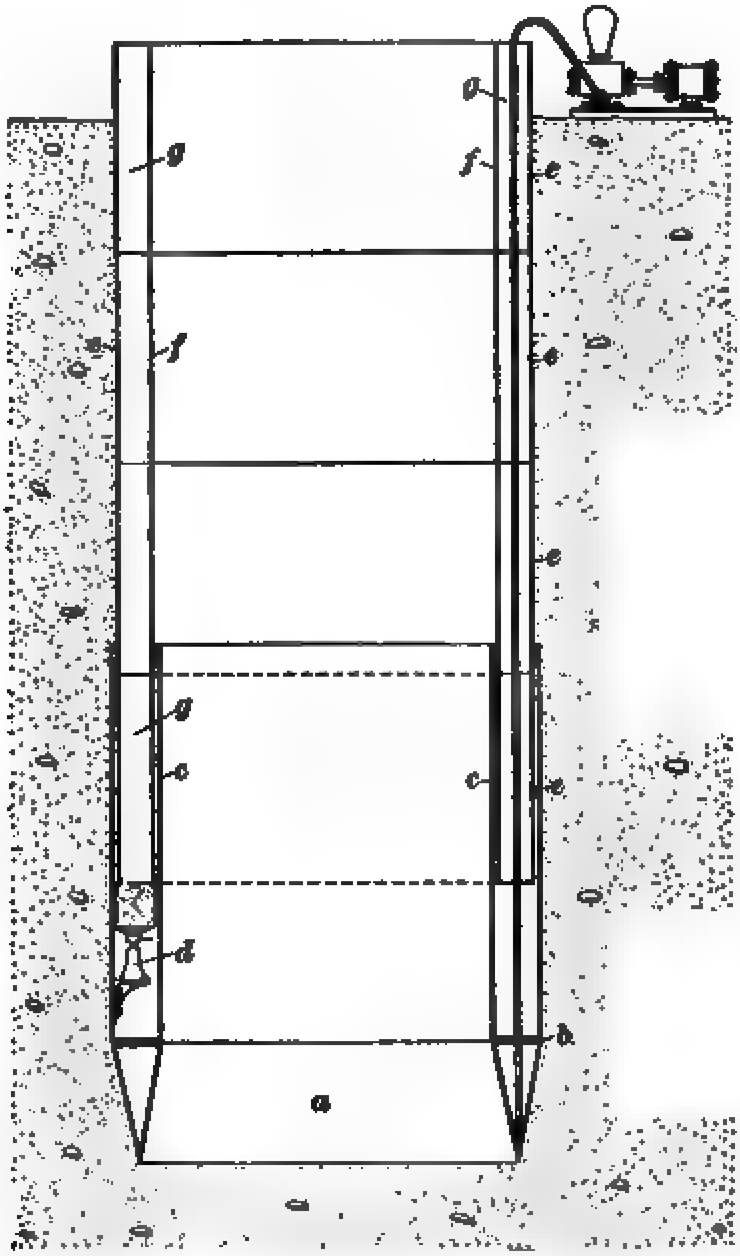


FIG. 34

blocking them against the lining *e f*, which extends to the surface and affords sufficient weight to push the shoe downwards, particularly when water is pumped through the shoe into the ground.

39. Sinking Shoe.—Fig. 35 shows in plan and elevation a simple form of shoe that will prove satisfactory where the

ground to be passed through is of the same consistency. It cannot be depended on, however, to pass through sand and then clay or hard pan to reach solid ground. This shoe measures 48 inches in height, and consists of iron plates that are riveted together and braced. The lower edge is beveled so as to act as a cutter, and the upper edge is about 18 inches above the I beams and cross-braces. The timbers that line the shaft rest on these beams and braces, and the 18-inch

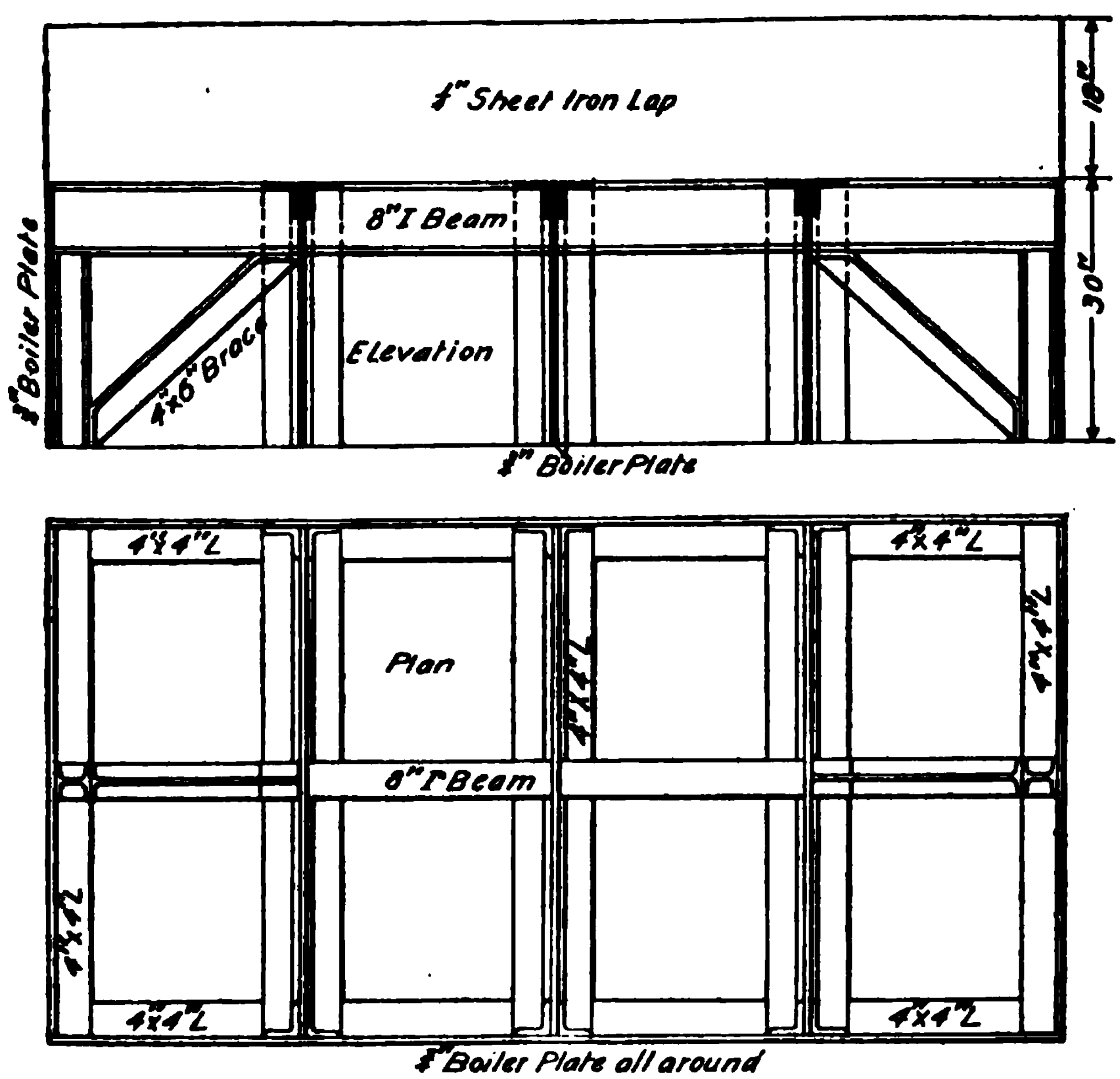


FIG. 35

space mentioned prevents them from moving out of line with the shoe. As the material is excavated from beneath the shoe, its weight permits it to sink; but in case it binds, jack-screws are applied to force it down.

40. Method of Using the Sinking Shoe.—Fig. 36 shows the method of using the shoe shown in Fig. 35, and also the method of lining the shaft. At the top of the shaft there is a mud-sill on which two shorter sills rest, and these three sills are bolted together, so as to have sufficient strength

to hold the weight that is suspended from them. The object desired is to prevent the shaft lining from getting out of plumb if the shoe moves downwards unevenly. At distances of about 10 feet, buntons are put in, and iron rods are screwed to them. The reason for having the 18-inch shoulder on the top of the shoe is to permit the shoe to be lowered sufficiently to insert these buntons. It will be observed that

Fig. 36

every other side plank for lining differs in width, an arrangement that aids in keeping the shaft plumb, since the sand packs in the spaces, particularly when a pump is used. The shoe is attached by chains to a swivel nut threaded to take a feed-screw hung from the lowest bunton. To make an advance, the feed-nuts are unscrewed just enough to permit a plank to be inserted under the lowest one for lining, and this is continued all around the shaft, blocking being inserted

to hold the planks above while the new planks are being placed.

In case the quicksand flows into the excavation, the shoe must be covered above the I beams and a pump pipe extended below the platform thus formed. The pump will probably perform the excavation in this case, so that the lining can be carried downwards in the same manner as before. The same system has been used effectively in ground free from boulders and clay; for instance, in one case a sand bed 50 feet thick was encountered 60 feet below the surface.

41. Weighting Shaft Linings.—Before the system of adding the lining on the shoe was adopted, the plan was to add the lining at the top and sink the whole shaft by jack-screws weighted at the top. This system could be followed only to a small depth, since the sand would pack about the lining and by pressure prevent any downward movement. It was also a difficult matter to keep the shaft lining plumb, since there was no way of regulating the direction of the shoe except from the top of the shaft.

42. Sinking the Susquehanna Shaft.—The Susquehanna shaft located at Hibbing, Minnesota, was sunk through 57 feet of dry sand, 50 feet of quicksand, and 31 feet of hard pan, or a total of 138 feet. Anticipating quicksand, a small shaft was sunk with the expectation of draining the surrounding territory and thus make it less difficult to sink a large working shaft 40 feet away. The inside dimensions of the small shaft were 8 ft. × 10 ft., and the shaft timbers were 12 in. × 12 in., separated by posts 3 feet long and suspended by hanging bolts $1\frac{1}{2}$ inches in diameter from shaft-timber bearers that were 24 inches in diameter and 32 feet long. At a depth of 45 feet, six $\frac{3}{4}$-inch-diameter wire ropes were used to hang the shaft timbers from the bearers, using 8″ × 8″ washers for the bolts. At 57 feet, when water and quicksand were met, there was considerable pull on the ropes, and two trusses were built to span the shaft collar, making the shaft-timber bearers the lower chords of ordinary queenpost trusses. From this depth, a shaft set

was beveled to an edge flush on the outside of the shaft, placed in position, and forced down with jacks like a shoe, filling in other timber sets above it. The timbers were then bolted together, after which water was pumped and sand excavated and hoisted.

At a depth of 85 feet, the pressure from water and sand being very great, the size of the shaft was reduced to 6 ft. × 8 ft. by building a solid box with the bottom edge beveled to the outside. This box was forced down to the hard pan and suspended from the trusses at the shaft collar by ten wire ropes 1¼ inches in diameter. During the entire process, a steel box 3 ft. × 5 ft. and 10 feet long was forced down in the middle of the shaft, to serve as a sump for the pump. The progress in the quicksand was very slow, varying from 6 inches to nothing a day, and work had to be carried on continuously by 8-hour shifts in order to hold the headway made.

The great danger in such ground lies in the sand and water bubbling through under the bottom timbers, and to prevent this, a plank or shovel was placed where the bubbling occurred, and the water was allowed to rise a few feet in the shaft and thus permit the sand to settle. To avoid cavities behind the lining, hay was rammed behind the timbers, and no more sand was hoisted than could be avoided, for if removed too fast, the sand outside the shaft would become loose and run and cause such great strains as to break the irons and ruin the whole work irreparably. In some cases, the pumps had to be stopped to avoid this running, which was always preceded by the bubbling mentioned.

43. This drainage shaft being completed through the hard pan to solid rock, the three-compartment hoisting shaft, 6 ft. × 16 ft. inside, as shown in Fig. 37, was started, with the usual square-set timbering and lagging to the quicksand. Here it was found that the drainage-shaft pumps had drained the ground at the hoisting shaft to such an extent that the water could be readily handled by a No. 7 steam pump. The same difficulties were encountered as at the drainage

shaft, but not to such an extent. The method of sinking was as follows: A drop section *a* of shaft timbers was framed and placed in position to act as a shoe. This section was the full size of the shaft, and was made of four sets of 12″ × 12″ timbers bolted together, the two lower sets being beveled as shown and shod with a shoe *b* of ⅛-inch steel plate. The cutting edge of the shoe had a 3-inch face, and the steel plate was shaped and spiked to the lower timbers. On the outside of this drop section was bolted a sheathing *c* of planks that had their lower edges mitered so as to offer less resistance to the sand. The upper ends of these planks extended above the drop section to the permanent timbers *d*, as shown, so that when the jacks *e* forced the shoe and drop section down, another permanent timber set could be put in and the quicksand kept out. Thirty jacks 2½ in. × 18 in. were operated between the drop section and the last permanent timbers, and when they had forced down the shoe a sufficient distance, they were removed temporarily and a new set of permanent timbers put in place. As it was necessary to keep this shaft vertical, extreme care was

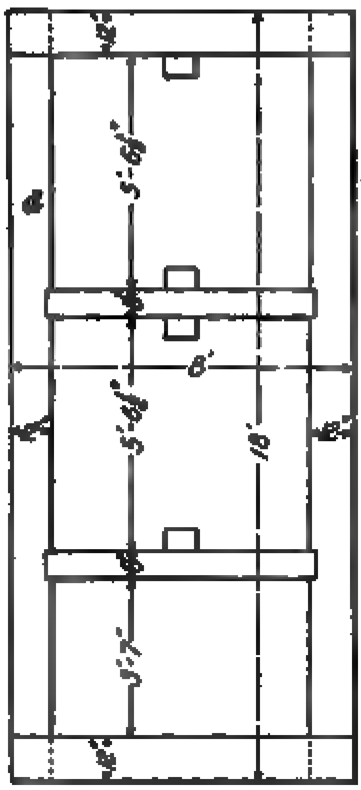


FIG. 37

exercised in using the jacks. The rate of progress through the quicksand was about 9 feet per month.

44. Shaft Sinking by Interlocking Channel Bars. A new system of sinking shafts through quicksand has been introduced with success near Johnson City, Illinois, where, at a depth of 65 feet, a stratum of quicksand 10 feet thick is encountered, which is followed by 4 feet of clay, then 18 inches more of quicksand and 6 inches of blue clay, and then shale. When drained of water, the sand sets hard, and timber is forced through it only with great difficulty. To overcome this, interlocking channel bars 12 feet long and in the form of a rectangle were driven down until they penetrated

FIG. 38

about 2 feet into the shale. In Fig. 38 is shown a section of this shaft lining with the different members assembled in the relative positions they occupy when placed in a shaft, while in Fig. 39 are shown the details of construction. The lining forming the sides is made up of 12-inch 33-pound channel bars *a* and *b* alternately arranged as shown, having riveted to them $4'' \times 4'' \times \frac{3}{8}''$ **Z** bars *c*, the latter being so placed as to engage with and interlock the edges of the plane channels. The corners of the shaft lining are made of 3-inch angle bars riveted together so as to interlock with the sides. If necessary, heavier channel bars can be used,

but in this case they were not needed. Channel bars thus arranged can be driven to varying depths by fitting in pieces from the top and driving the preceding one down ahead. The channels interlock nearly water-tight, and by cementing above and below them when in a permanent position, the water may be practically shut off. The additional angle bars shown at *b*, Fig. 39, are for the purpose of preventing the **Z** bars from separating when under pressure.

To use this system, the shaft should be started 2 feet larger each way than the size desired, and should be sunk

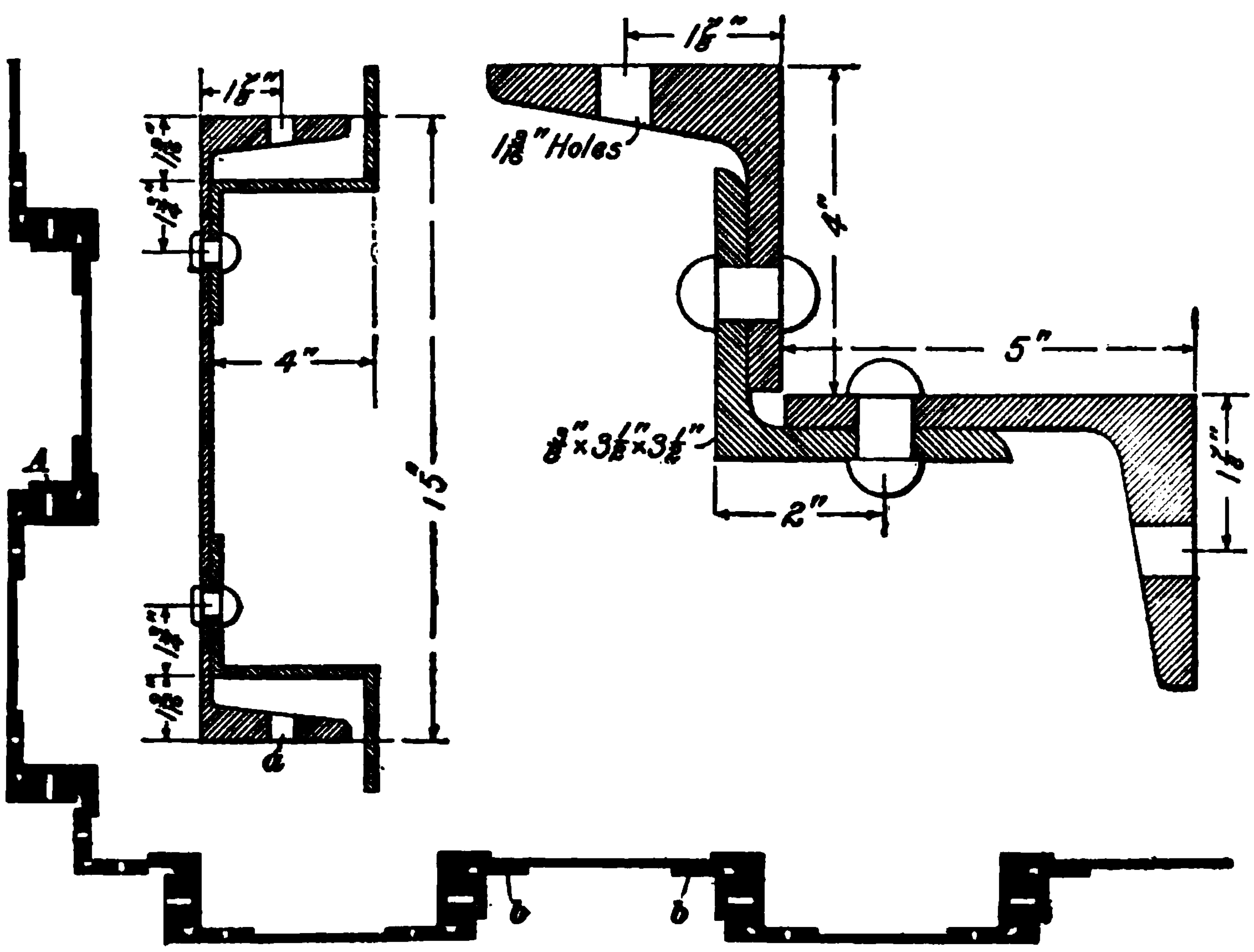


FIG. 39

with this cross-section to the quicksand to be encountered. The channels are then driven down inside the shaft without removing any of the sand. Thus, the channels take up about 5 inches, and, allowing 6 inches for the timbers, there will be only 1 inch extra space on each side. With timbers 6 inches thick, therefore, an 8′ × 16′ shaft would be the maximum size that could be obtained from one started 10 ft. × 18 ft. in section. Before hoisting any material, the channels must be set plumb in a solid frame and driven

square through the sand to solid rock, and at no time should any channel be driven more than 2 feet ahead of the others. While driving, a perfectly fitting head is placed on the top of the channel bar to protect it from being broomed. The holes *a* on the vertical edges of the channel bars are for the purpose of splicing the channels together after they are driven into place. The advantage of this system is that no sand has to be taken out until the channels are driven clear through and embedded in some firm material, thus balancing the pressure of the material, which would bind the lining if it were all on the outside. The price of this lining is about $2.50 per square foot.

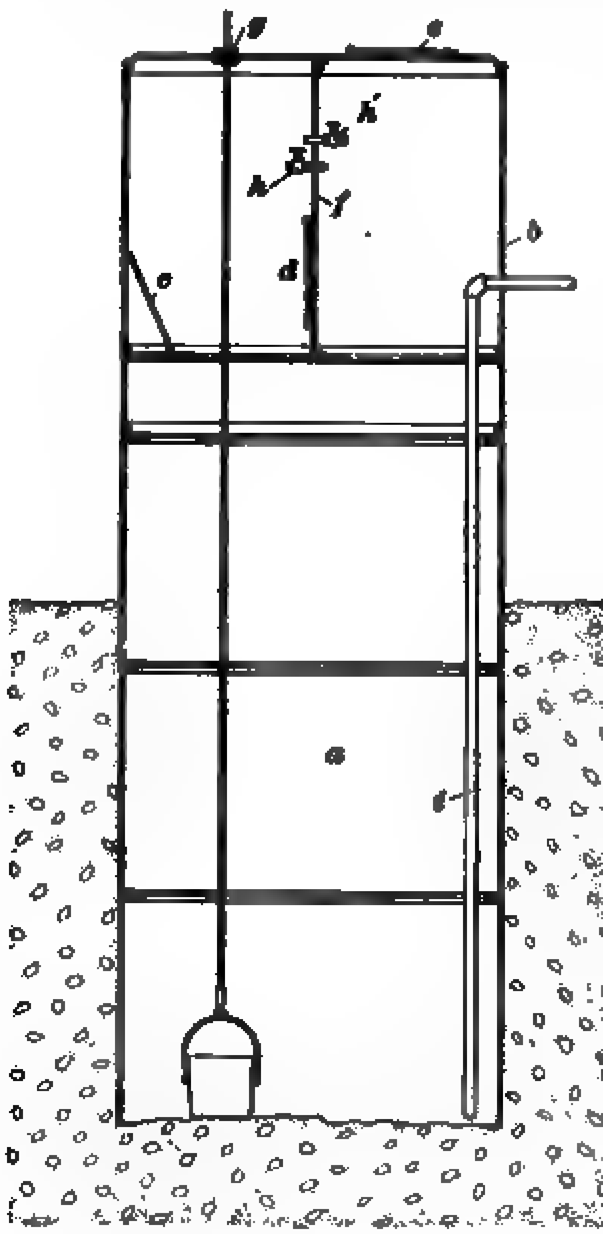


FIG. 40

45. Pneumatic Method of Shaft Sinking.—The pneumatic method of shaft sinking can be employed in some cases, but it is generally adopted to obtain solid foundations for bridges or tall office buildings. The objections to its use are that the men are obliged to work under a pressure of air sufficient to keep the material from running into the excavation, and this pressure increases with depth. Fig. 40 is a cross-section of an iron-plate caisson *a* surmounted by an air lock *b* made of boiler plate. In order to keep the air in the caisson and permit the excavated material to be

raised, there is a stuffingbox *g* through which the hoisting rope passes, also a partition *f* and a system of doors *c*, *d*, and *e*. The workmen use the doors to go into and out of the caisson. The material to be removed is hoisted through the door *c*, which is closed as soon as the bucket reaches the air-lock platform, the other doors being closed during the operation. The valve *h* is next opened to equalize the air in the two compartments and to permit the door *d* to be opened, after which the bucket is transferred to the other compartment; the door *d* and the valve *h* are then closed and the door *e* opened, through which the bucket is hoisted, dumped, and returned to the air lock. The door *e* is next closed and the valve *h'* opened to equalize the pressure, when the door *d* is opened and the bucket transferred to the other compartment, after which the valve *h'* and door *d* are closed. Air is admitted into the compartment containing the bucket; then, the door *c* is opened and the bucket lowered into the caisson. In order to remove the water through the pipe *i*, a pump may be needed, and in very soft material this may be all that is required for excavation, provided the men work the material from under the caisson edge so that it may sink uniformly. Air is let into the caisson by a pipe similar to *i*.

There is the same objection to this system as to others where gravity is depended on to sink the lining; namely, that it can only penetrate the ground a short distance before the outside pressure of the ground holds it like a vice, necessitating the use of hydraulic jacks. It then becomes only a matter of time when the use of jacks will prove futile in forcing the lining down.

The expense connected with the pneumatic method of sinking is not so great as might be supposed, but there are several disagreeable features that have militated against its adoption for mining purposes. The men in the caisson can only work short shifts, even when the gauge pressure is but 25 pounds per square inch; above this pressure, they are subject to vertigo, nosebleed, intense pains in the back, or paralysis. These effects seem to be felt more on coming

out than when in the caisson, and may be considerably lessened by gradually decreasing the pressure in the air lock and remaining in that atmosphere for some time before going into the open air.

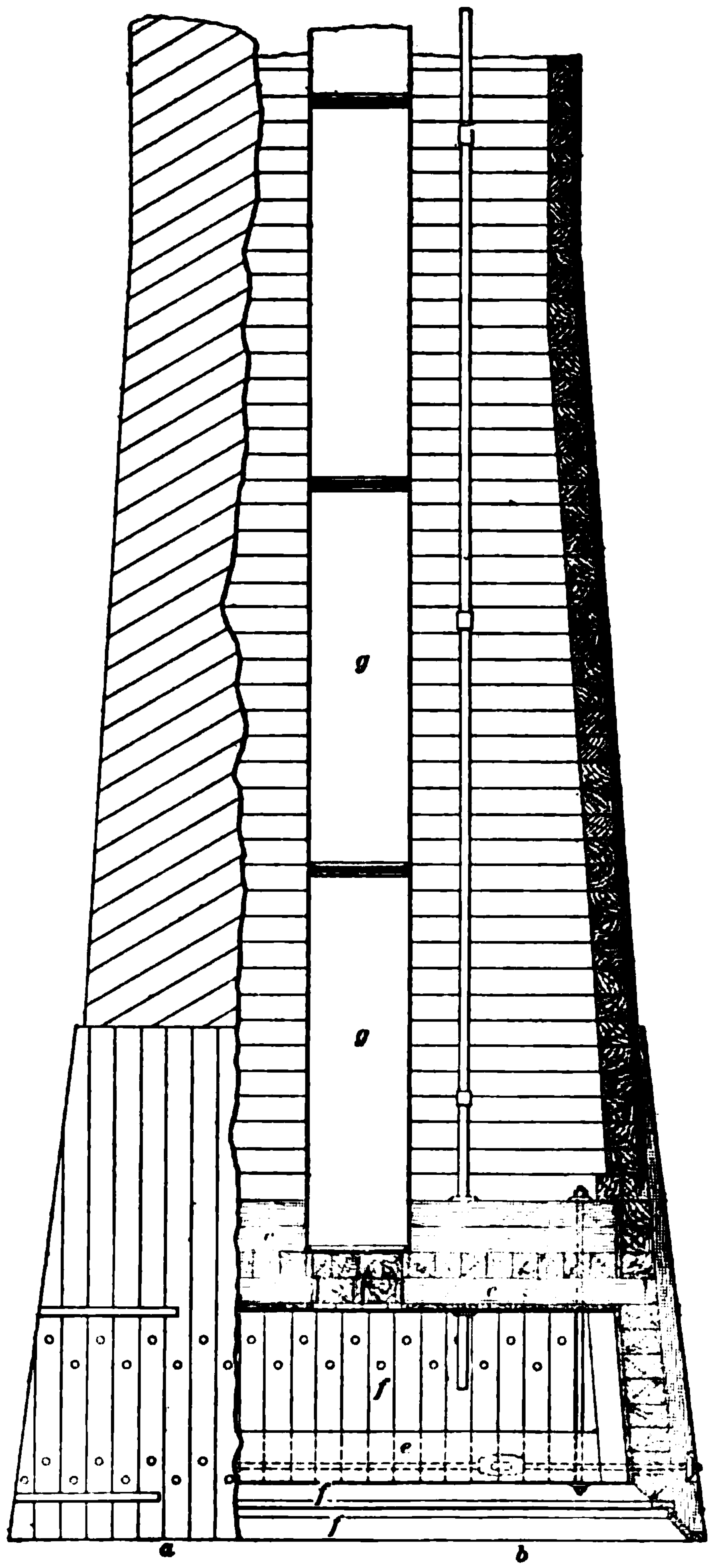


FIG. 41

46. Shaft Sinking With Timber Caisson.—Shafts have been put down to bed rock by using caissons similar to those employed in bridgework, but there is difficulty in making timber caissons sink in soft ground, even when weighted and jacked down. Fig. 41 illustrates the wooden caisson and air lock. In the illustration, *a* represents the outer side of one end of the caisson and the position of the plank sheathing, while *b* shows in section the air-shaft *g*, the roof of the caisson *c*, and the heavy timber braces *e*, which are placed across the caisson to keep the edges in place.

Chamber *f* at the bottom of the shaft is the caisson in which the men work, and it is kept free from water by air pressure. The roof of the caisson *c* in this case is constructed

of four layers of 12″ × 12″ timbers laid in tar, pitch, or cement. Inside of the roof there is a 3-inch plank lining, and all the joints are rendered air- and water-tight by calking. A shaft or tube 4 feet in diameter is carried up through the center of the shaft, as shown at *g*, and the air lock is attached to and forms a part of the top of this shaft, pressure tube, air-shaft, or main shaft, as it is called. Two 4-inch pipes are carried to the caisson chamber, one for supplying fresh air and the other for conveying the excavated material out of the shaft. The shaft is sunk to the water level without the use of compressed air, after which the caisson is put in place, and the shaft lining, which is composed of cribbing, is commenced on the top of the caisson and carried up as the work progresses. The entire structure, lining and all, sinks with the caisson. A sufficient pressure of air is introduced into the caisson chamber to keep back the water, and men enter it through the air-shaft *g*, which is provided with an air lock at the top.

47. The Air Lock.—The air lock consists essentially of a chamber having two doors, both opening toward the caisson. As a man comes to it from the outside, the outer door is open, and he enters the chamber. After closing this door, he allows the compressed air from below to enter the chamber through a pipe and valve, and when the air in the chamber has reached the same pressure as that in the tube, the door leading to the tube is opened and he descends to his work by means of a ladder through the tube and into the caisson.

48. Position of Air Lock.—The position of the air lock in regard to the caisson chamber is a point that has been discussed a great deal. If it is placed at the bottom of the air-shaft, the men do not have to climb the ladders under air pressure, thus saving a great deal of very fatiguing work. On the other hand, if a sudden inrush of water occurs through a break in the side of the caisson or in the air pipes, it may be impossible for the men to escape through the air lock before the caisson is filled with water, and in

such a case they would be drowned in the caisson chamber. If the air lock is at the top of the air-shaft, the men can escape into the air-shaft, and thus be safe from the rising water. This matter has been compromised by placing the air lock in the shaft at a point about half way between the caisson and the top of the work when finished. In the caisson illustrated in Fig. 41, the air lock is placed at the top and forms a part of the air-shaft.

49. Removal of Excavated Material.—Originally, the material excavated from the caisson had to be hoisted through the air-shaft and stored in the air lock until this was filled; then, the lower door connecting with the caisson was closed, the outer door opened, and the material discharged.

50. Auxiliary Air Lock.—The method of removing excavated material just mentioned was extremely expensive and led to the invention of an auxiliary air lock on the side of the main lock, into which the material could be placed. When this lock was filled, it was discharged without affecting the pressure on the main lock. This was accomplished by having the doors at the two ends of the auxiliary lock so constructed that they could not both be opened at the same time, and after the material had been placed in the lock, the inner door was closed and the outer one opened. The general principle of this method is illustrated in

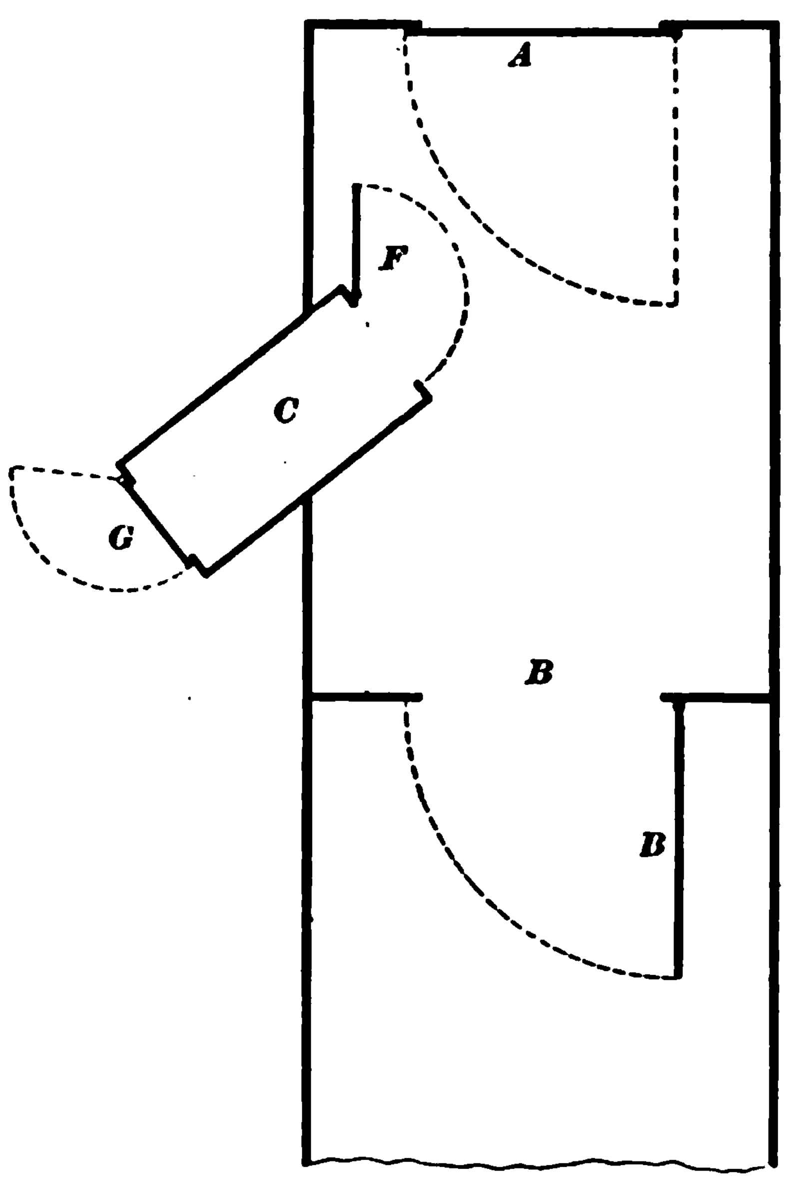


Fig. 42

Fig. 42, where B represents the door leading from the main air lock into the air-shaft, A the door leading from the

atmosphere into the main air lock, and *F* and *G* the two doors of the auxiliary air lock. When in use, the doors *A* and *G* are closed, and the doors *B* and *F* are open. Material is hoisted into the air lock and dumped into the auxiliary lock *C*. When the auxiliary lock is full, the door *F* is closed, the door *G* opened, and the material slides out of its own accord. This device has greatly cheapened the pneumatic method of sinking, but it entails a great amount of labor in hoisting the material into the lock.

51. Aspirator.—The next improvement was the invention of an aspirator, by means of which the fine material could be blown or aspirated out of the caisson. The procedure followed was to mix the sand and gravel with water and feed it into a pipe connected with the atmosphere, the mixture being then blown out in a stream by the air pressure in the caisson.

When only a few boulders are encountered during the sinking, they are carried down in the caisson, and if the opening is a shaft, are removed and hoisted out after the roof of the caisson has been cut away. If there are a large number of boulders present, however, it will be necessary to blast them and hoist the pieces out through the air lock.

52. Shaft Lining.—The shaft lining as built on the top of the caisson, Fig. 41, is composed of timber and is built up as cribbing. The outside of the lining was covered with 3-inch plank lagging placed diagonally, as shown at the left of the illustration.

FREEZING METHOD FOR SINKING IN QUICKSAND

53. Principles Involved in the Freezing Methods. Pœtsch evolved the plan of freezing quicksand in such a way that excavations can be made in it the same as in solid ground. The Pœtsch system is based on the use of a freezing mixture that will absorb heat when brought in contact with the material to be frozen, and that, being much below the freezing point (32° F.), will eventually reduce

the material to such a temperature that it freezes solid. A solution of calcium chloride that will not freeze until the temperature is 50° F. below zero is employed for this purpose, and is pumped into a series of tubes sunk in the ground to be frozen. Calcium chloride in solution is a brine that does not freeze if cooled by means of ammonia gas, but if circulated through tubes properly arranged in the ground, will cause the earth that comes in contact with the tubes to freeze solid.

54. Conversion of Gas to a Liquid.—To convert a liquid to a gas, heat is applied; but a large part of the heat is latent, that is, does not appear as heat nor raise the temperature of the gas. Conversely, when pressure is applied to convert a gas into a liquid, the latent heat is liberated. Each gas has a particular pressure at which it may be converted into a liquid at a given temperature; ammonia, for example, becomes liquid under a pressure of about 90 pounds per square inch above the atmospheric pressure when the temperature is 50° F. If the temperature is not raised and the pressure is kept constant at 90 pounds per square inch, the ammonia will remain a liquid; but if the pressure is reduced or the temperature increased, the liquid will become gaseous.

In the freezing process, anhydrous ammonia gas is liquefied by compression in the pumps of an ice machine, and the liquid leaves at a temperature of 102° F. and passes through pipes surrounded by cold water to reduce the temperature. The liquid ammonia is then made to flow into a long series of pipes placed in a large wooden tank containing a solution of calcium chloride. The liquid ammonia expands into a gas in these pipes, and extracts heat from the solution surrounding them to such an extent that the temperature of the calcium-chloride solution is reduced to about 8° F. The ammonia gas is then returned to the compressor to be liquefied and utilized for the production of more cold. Calcium chloride, as stated, is capable of absorbing a large amount of cold without becoming frozen; consequently, if

the solution should leak into the material to be frozen, the desired object could not be accomplished.

55. Freezing the Ground.—To freeze the ground, a series of casing pipes are driven down or sunk by the rotary system to solid rock. These pipes are then cleared of material by means of sludge pumps and streams of water. The rock having been reached, a drill is used to bore into the rock for a distance of 4 or 5 feet. Pipes 5 or 6 inches in diameter are let down through the casing pipe, the lower pipe being plugged at its end either before of after sinking. The casing pipe is next removed, which leaves the solution pipe in contact with the earth to be frozen, as shown at *a*, Fig. 43. Inside of the pipe *a*, the solution delivery pipe *b*, 1½ or 2 inches in diameter, is lowered nearly to the bottom of the 5- or 6-inch pipe, as shown. The brine is pumped from the cooling tank of the ice machine into the small pipe connected at the surface with the pipe *c*, and is thus made to flow to the bottom of the large pipe, from which place it rises and passes back through the pipe *d* to the cooling tank, where it is again cooled and circulated as before.

In the figure, which illustrates the practice as conducted in sinking a shaft at Iron Mountain, Michigan, the pipes are shown in a circle about the ground to be frozen. They were placed about 3 feet 6 inches apart and formed a circle 29 feet in diameter. The ground was frozen solid to the center of the shaft and for a distance of about 13 feet outside of the tubes near the bottom. The frozen quicksand is said to have resembled a compact sandstone, and was excavated as though it were rock. The excavation was carried inside of the circle formed by the pipes, and the ground was supported by hanging timber sets from the top down. It is customary to make the first timbering temporary, and then build from the solid rock a strong lining that will support the pressure exerted by the quicksand after it has thawed.

56. Advantages of the Pœtsch System.—The many uncertainties connected with the other systems of sinking through quicksand are avoided by using the freezing method.

FIG. 48

The expensive and slow work of timbering and the vast amount of pumping needed in other systems are avoided. In France, shafts have been put down 300 feet by this method, and, since excavation when under way can proceed quickly, it is considered in some cases to have been cheaper than other systems.

57. Freezing by Ammonia Gas.—M. Gobert introduced a modification of the Pœtsch system that acts more quickly, since it brings the cold-producing substance in direct contact with the ground rather than through an intermediate solution of brine. In practice, the temperature of the sodium-chloride solution in the tubes seldom goes below zero, owing to the loss of cold due to pipe connections and circulation defects. When ammonia is vaporized and sent down the inner tube, a lower temperature is obtained, for the gas is capable of producing a cold of $-22°$ F. Although this temperature would not be realized in practice, owing to losses due to circulation and pipe connections, nevertheless, a lower temperature is available than when brine is used. After circulation, the ammonia gas is pumped back and liquefied, so that this process is practically as continuous as the brine process, and has the advantage of freezing the ground more quickly. It is also claimed to be more economical than brine, both in the outlay for the plant and the cost of operation.

58. Freezing Quicksand Below Rock.—When a shaft has been sunk in rock and then encounters quicksand, the tubes for freezing mixtures are put down at an angle from inside the bottom of the shaft. The pipes are then connected and carried to the surface, and in order to prevent a loss of cold, the pipes in the shaft are covered with a non-conducting material. The ground is then frozen, as already explained, either by the solution of brine or by ammonia gas. If a stream of running water is encountered, the process may not be capable of freezing the ground, since the water will continually abstract the cold, and before it can be congealed, will move away from the pipes.

SINKING IN SOLID ROCK

59. Excavating Apparatus.—If a shaft is to be sunk to a depth of 250 feet or more, and timber and machinery, such as pumps and drills, are to be handled in addition to hoisting the broken material, the work could be done with a horse whim; but this method would be so slow that a steam hoister would be more satisfactory and more economical, even if it were discarded for a more powerful hoisting engine, after the sinking was completed. Where a hoister is used, there must be a head-frame, an engine, and a boiler house. A 30-horsepower engine with a 60-horsepower boiler may be employed, and the balance of the boiler power utilized to run a steam pump. If air drills are required, more boilers, as well as an air compressor, will have to be added to the plant. Three iron buckets should be provided, so that if one of the buckets gets out of order while hoisting material, it can be replaced. A blacksmith shop with forge for sharpening drills, making spikes, and repairing tools should also be included.

FIG. 44

60. Head-Frames.—The head-frames employed for sinking purposes are usually temporary affairs, and are therefore not so expensive nor so elaborate as permanent head-frames. In many instances, however, the head-frame

used for sinking has been retained as the permanent head-frame, and there is no valid reason for not doing so whenever the purpose for which the shaft is to be used has been determined by exploration—but not otherwise. Fig. 44 shows a common form of gallows frame used for sinking purposes. The platform *a* has a door *b*, which is closed when the bucket is dumped, so that no material will fall down the shaft, but at other times is often left open. The object of the raised platform is to permit the bucket *c* to be dumped directly into the rock car *d*, and thus avoid extra handling.

Another type of head-frame for sinking is shown

FIG. 45

in Fig. 45, and is preferable in some respects to the gallows frame. The platform *a* may be arranged so that the ore can be dumped over the side of the platform, or by keeping the doors *b* closed, the bucket may be dumped directly into a car that is run on to the platform.

61. Landing Doors.—Several complicated doors for landing platforms have been designed to open automatically when the bucket reaches them, but all such arrangements are dangerous, inasmuch as they are liable to become broken and thus tip the contents of the bucket on the heads of the sinkers. As there must be doors at the landing, the best for the purpose are those that can be opened and closed by hand, and in order to operate them without too much exertion, they are usually balanced by weights. Fig. 46 (*a*) and (*b*) shows a plan and an elevation of double doors that are easily opened

and closed by a lever *a* connected by reach rods *b* and *c* with two bell-cranks *d* fastened to shaft *e*. To this shaft the hinges of the doors *f* are keyed, as shown in the plan, Fig. 46 (*b*). The ends of the bell-crank *d* are prolonged and counterbalanced by the weights *g*. It will be noticed that the shafts *e* extend across the opening, and that there are four weights and four bell-cranks. When it is desired

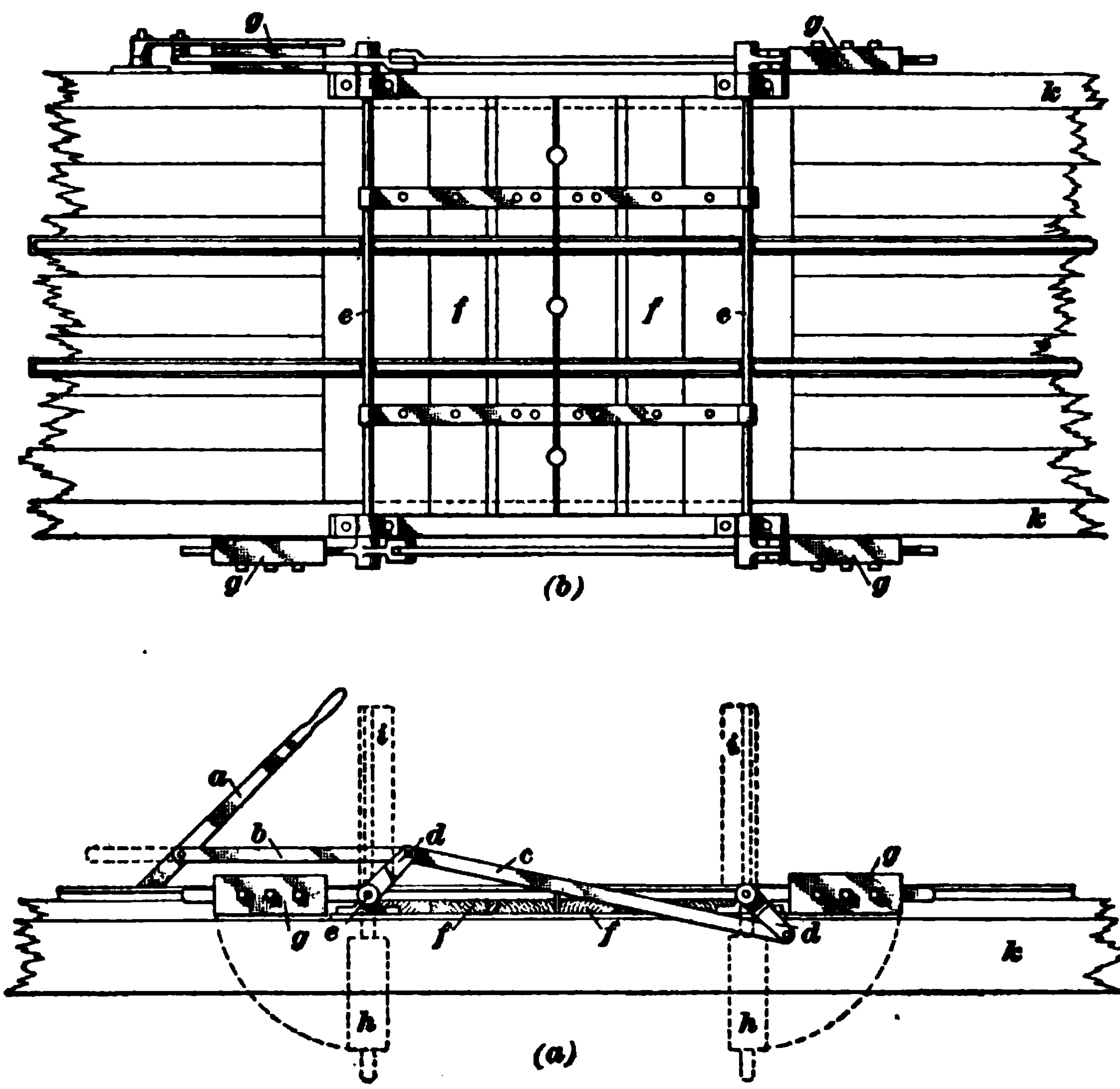


FIG. 46

to open the doors, the lever *a* is pulled backwards, and this throws the bell-cranks with their balances, as shown at *h*, thereby opening the doors, as shown by the dotted lines *i*. These doors are supplied with rails for the landing car to cross, and they rest on heavy longitudinal beams *k* that pass over the opening.

62. Buckets.—The buckets used most for sinking purposes are made of iron, and are capable of holding from

6 to 12 cubic feet, or ½ to 1 ton of rock. Buckets similar to that shown in Fig. 47 are made from 16 to 28 inches in diameter at the top, from 14 to 28 inches at the bottom, and vary in height from 26 to 38 inches. The weight, of course, depends on the size, and is from 180 to 470 pounds, while the cost is usually from $18 to $50, the price for such work being about 10 cents per pound. The bail *a* is attached below the center of gravity of the bucket, so that the tendency of the bucket is to turn upside down; this, however, is prevented by a link on the bail, which slips over a short pin riveted to the bucket, as shown. One pin and link, however, is not enough for heavy buckets, and there should be two.

63. Bucket Riders.—As the tendency of buckets to swing and of ropes to twist increases with the depth of the

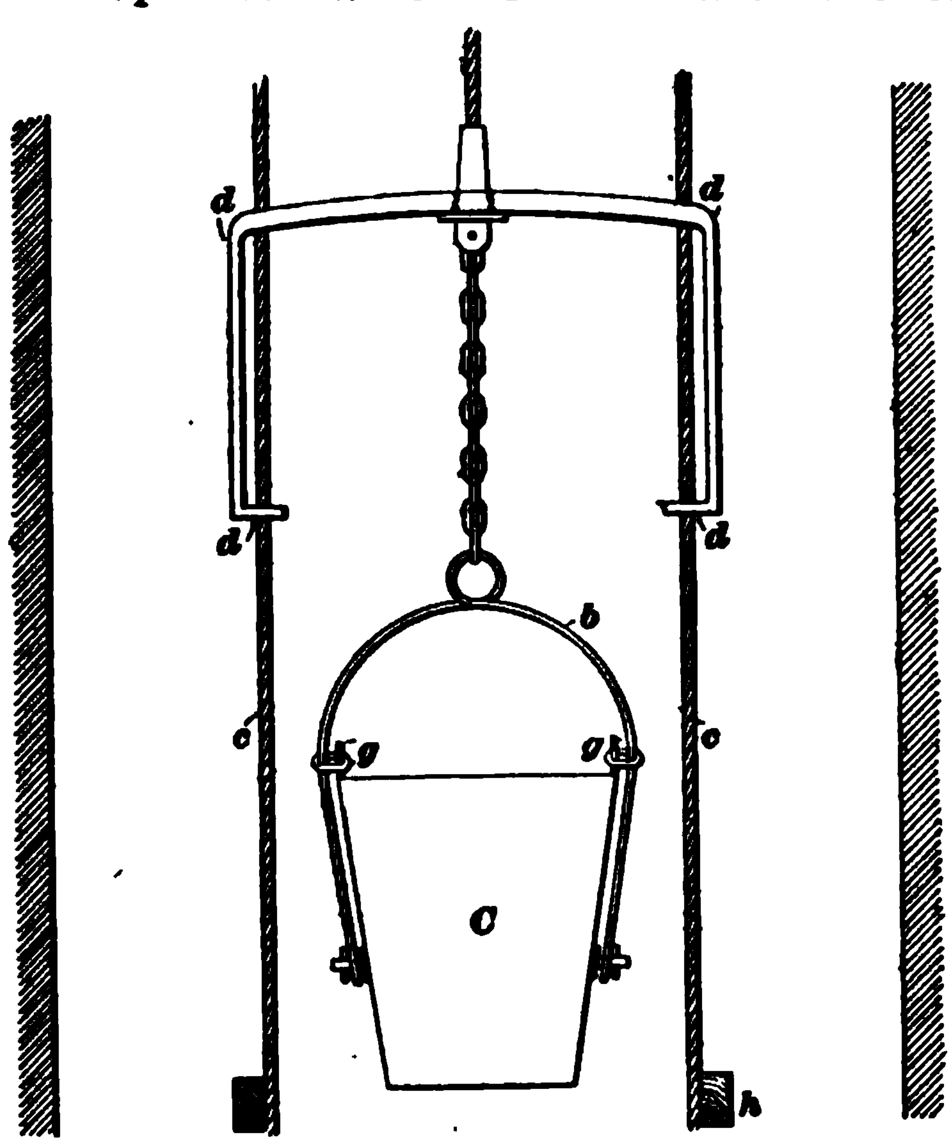


FIG. 47

shaft, it is a good plan, in order to make the bucket travel vertically, to use guide ropes when hoisting from deep shafts. Fig. 47 shows the arrangement of guide ropes *c*, which, when so used, are suspended from the head-frame

over pulleys and fastened to the floor at the lower end of the shaft by means of iron rings. At the end of the rope passing over the pulleys, heavy weights are suspended to take up all slack and make the ropes taut. The bail *b* of the bucket *C* is fastened to a rope, which is held central by the crosshead *d*. The crosshead is supplied with arms, or riders, and may be stopped wherever desired in descending, and thus kept out of the way of the loaders. When the crosshead is stopped, the bucket may continue its descent, as the hoisting rope passes through the crosshead of the rider; but on its upward flight, the bucket picks up the rider by the piece of iron attached to the rope socket, and is thus kept from swinging. The bucket is kept upright by the rings and pins *g*.

64. Dumping Buckets.—If a raised platform is used for a landing, the contents of sinking buckets can be dumped over the side of the shaft, either on the ground or into rock cars. A bucket can be dumped over the side of the head-frame by means of a short rope suspended from the top of the head-frame and having a hook that is coupled by the topman to a ring in the bottom of the bucket. When the hoisting rope is lowered, the bucket is prevented from descending by the hook and rope attached to it and turns upside down, at the same time swinging clear of the landing and thus dumping over the side.

When a short rope is used, the buckets have their bails fastened to the top; hence, their center of gravity is not below the bail fastening, and they can be dumped by lowering them and at the same time swinging them away from the shaft opening. With similar buckets, the topman can push them until they strike a rail, and then, as the rope slacks, they will upset. This latter method requires more manual labor, but is preferred by some, since the dumpsman is obliged to close the door over the shaft before dumping the bucket.

Buckets with their bails hung below the center of gravity could be dumped over the side of the landing, but it is more

general to use such buckets where the rock cars are run over the shaft; in fact, there is little excuse for their employment in any other situation. Fig. 48 shows the plan of a

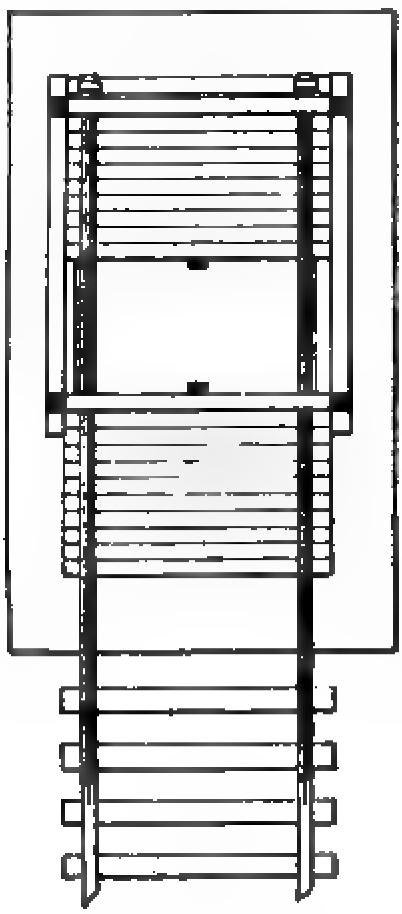


FIG. 48 FIG. 49

landing with tracks crossing over the shaft. Doors are placed in between the tracks, to prevent the material dumped from falling down the shaft. Fig. 49 shows a side elevation of a head-frame, with a bucket, a rock car *e*, and a hand rail *f*.

65. Shaft Coverings.—In sinking operations, doors made of planks that will support the weight of at least three men should always be placed on the landing, and the landing itself should be covered with stout 3-inch planks resting on suitable floorbeams so as to make a firm floor. It is not necessary to board up all around the landing, but the precautions should be such that nothing can fall down the shaft. There must be openings left for air, but they should be under the platform, since the doors must often be kept closed, except when hoisting, and the platform at all times must be kept clean and free from loose pieces of rock.

REMOVING WATER FROM SHAFTS

66. Sinking Pumps.—In shaft sinking, the shaft may possibly be kept clear of water by bailing it into the ore bucket, but in most cases this method will not meet the requirements, and special arrangements must be provided. Sinking pumps are made portable, so that they can be lowered and raised, and are supplied with eyebolts, so that they may be suspended from a wire rope. Since the pumps must support the weight of the water and be steadied while at work, they are also provided with wrought-iron hooks for attachment to the shaft timbers, as shown in Fig. 50. If there is any danger of flying rocks from a blast hitting the pumps, they must be uncoupled from the column pipe and raised to a place of safety. As this requires considerable time, it is customary, wherever possible, to shield the pump with a battery made of timbers or a rope mat, which is not difficult to accomplish if mats are used to blast against. The tail-pipe of the pump must be raised before each blast, and if long, should be disconnected from the pump. If the column pipe is long and is disconnected from the pump, all the water it contains, unless it has a check-valve, will flow back into the sump. At times, however, the pump must be disconnected from the water and steam pipes, as the sinking

FIG. 50

progresses, and placed at a lower level, thus making it necessary to add to the length of the column pipe, and to provide a water gate in the pipe in order to prevent the water contained therein from escaping into the shaft.

67. Water Rings.—When water-bearing strata are penetrated and the water cannot be dammed off, a water ring, such as that shown in Fig. 51, may be made by widening out the shaft at *a* and then contracting it at *b*. From this ring, the water may be conducted to the sump below or to tanks. In case there is considerable water collected in this way, a tank hung in the shaft, in connection with water buckets, may be used to advantage.

FIG. 51

68. Water Buckets for Sinking.—Fig. 52 (*a*) shows the side elevation of a collecting tank, with buckets *b* in section and elevation above. Fig. 52 (*b*) shows an end elevation of the tank and a section through the buckets. The tank is suspended from crab winches at the surface by wire ropes, which pass beneath the tank in grooved pulley wheels *c*. This arrangement permits the tank to be lowered as sinking progresses or raised when blasting is carried on. The ropes that support the tank are guides for the bucket crosshead *d*, hence they run true and enter the tank in their downward flight. The bottoms of the buckets are supplied with valves *e*, which open upwards when lowered into the water, and close from the weight of water when raised. To the valves are attached reach rods *r*, which are also pivoted at one end to levers *l*. When the bucket reaches the surface, the levers *l* strike against suitable stationary objects, which cause them to lift the reach rods, and the valves consequently discharge the water automatically. Hoisting engines are used for raising the water buckets,

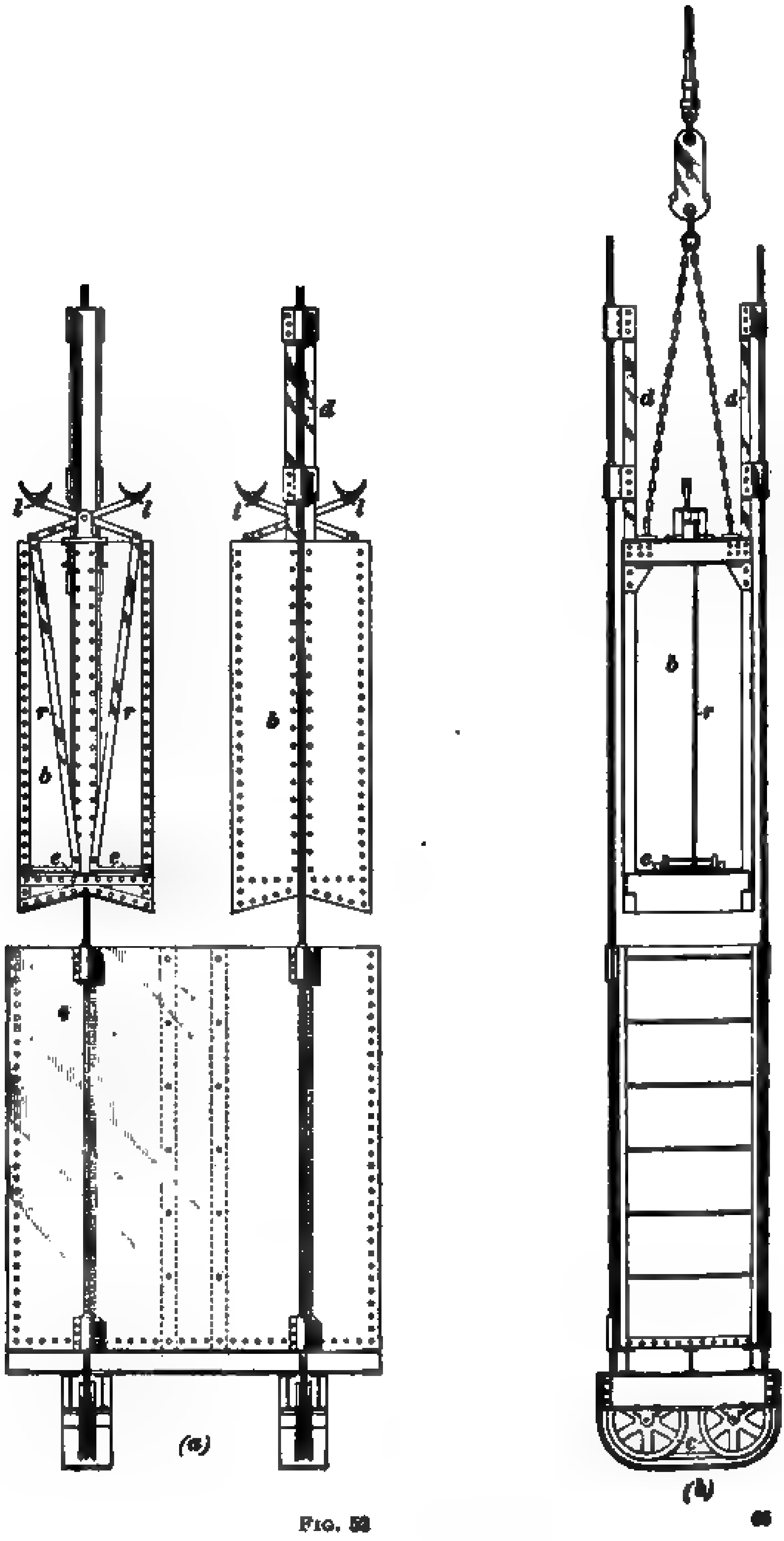


FIG. 58

FIG. 58

and in deep shaft sinking, this system is very effective; besides, it possesses an advantage over large sinking pumps in not having to deal with pump pipes.

69. Lodgments.—As a rule, the water met with in sinking comes from various horizons that are water-bearing or are fissured formations. If the strata below such formations are not fissured, it may be possible, by means of lodgments, or sumps, to hold the water at this point and prevent it from going to the bottom of the shaft. In deep shafts, this method is important, since water nearly always interferes with the work of sinking. Fig. 53 shows a lodgment *a* from which the water is drawn off to a pump sump situated on level *b*. The lodgment is made water-tight with bricks and cement mortar, but can be entered if necessary through the opening *c*.

SHAFT LININGS

MASONRY SHAFT LININGS

70. Concrete Linings.—The timber in shafts generally has to be replaced from time to time, and besides this is liable to catch fire; consequently, where a shaft is to be used for a number of years, it is considered good practice to make the lining of masonry. As most of the shafts in the United States are rectangular and masonry is not suited to that shape, it has not been customary to use such linings; however, since concrete makes an effective and durable lining in some instances, it is rapidly coming into use for that purpose.

71. Concrete-Block Lining.—If arrangements are made to drain the water that accumulates back of a lining and thus creates a heavy pressure, cement blocks may be used for shaft linings. For this purpose, in water-bearing strata, drain pipes with cast-iron water rings at intervals of about 20 feet are placed back of the lining. Fig. 54 shows a lining of cement blocks *a*, which are made with a groove *b*

on the under side and a ridge *c* on the upper side. These grooves and ridges correspond with mortises and tenons in timber, and are so arranged that the adjacent blocks will fit together as they are put in place. The blocks, which are of large size and weigh nearly a ton, are made at the surface, allowed to harden, and are lowered into the shaft as needed. Cement mortar is spread over the top of a block just before the next one is put in place, and thus forms a practically solid and waterproof lining that becomes stronger with age.

The curb *d* for carrying the water can be made of either cast iron or wood, or even a cement block with an iron drain pipe can be used. In the illustration, a flat, cast-iron curb is shown, having a hollow *e* through which water is drained from the pipe *f* to the pipe *g*. The pipe *f* can be packed all around with fine, loose, broken stone, which will thoroughly fill the space between the rock and the lining, and yet furnish interstices through which the water can drain to small holes in the pipe. Another method that could be employed in case there was only a small amount of water would be to drill holes into the side walls for nipple pipes *h*, and after the nipples were screwed into pipe *f*, make the concrete solid between the blocks and the rocks. The curb *d* is made in sections and is connected with drain pipes *g* so that the water will flow to the nearest sump.

FIG. 54

72. Expanded-Metal and Concrete Linings.—Concrete blocks are sometimes made with a thin piece of crimped metal placed in them, which acts as a binder and stiffener. A number of shafts in Pennsylvania have been relined with concrete made of one part of cement, two parts of sand, and five parts of broken stone not greater than

1 inch in diameter. The shaft about to be described was originally timbered with an inner and an outer plank lining, the space between the planks being puddled as shown in Fig. 31. The inner lining and the puddled clay were removed, but the other lining was kept in place in order to hold back the loose walls. This caused a variation in the concrete lining, amounting in some cases to 22 inches, so that the

Fig. 55

thickness of the concrete varied from 8 to 30 inches. When a section of old lining and puddled clay had been removed, a box braced as shown in Fig. 55 was substituted. Between the old outer lining and the planks of the box, the concrete was rammed and allowed to set. The work of relining was commenced at the bottom of the shaft and carried upwards, the buntons being taken out and replaced by concrete, as shown in Fig. 56.

While the partitions between the compartments are not solid, they are continuous, as may be seen in Fig. 57, which is a sectional elevation of the shaft. The oval openings are

Fig. 56

left for the purpose of examining the cage guides and permitting the air to circulate when the cages are moving. The concrete is stiffened by $\frac{5}{16}$-inch rods *h*, which are bent into sharp angles at intervals, in order to permit the cement to obtain a better binding than would be possible if the rods were straight.

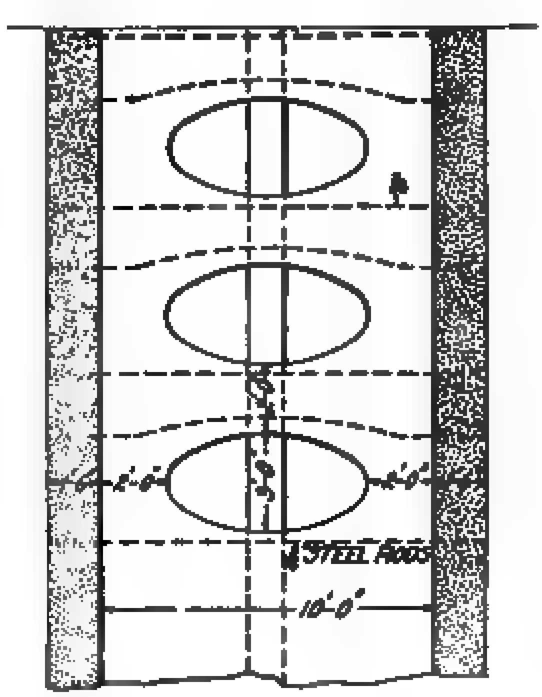


Fig. 57

The expanded metal, which is represented by dotted lines on the outside of the concrete, Fig. 56, is made in sheets from 6 to 8 feet wide and $\frac{3}{16}$ inch thick. It is merely corrugated sheet iron, which acts as a binder and stiffener for the concrete. When cement sets, it contracts, and the corrugations in the metal permit the iron to contract in one place and lengthen in another without injuring the cement.

73. The Richmond Shaft Lining.—Fig. 58 shows a plan, Fig. 59 a side elevation, and Fig. 60 an end elevation

of the concrete lining and false work of the Richmond shaft, near Scranton, Pennsylvania. Other illustrations of this shaft were given in Figs. 8 and 9. In putting in the concrete *a*, the false work was built up from below and was braced as shown. The posts *b*, with their 12″ × 12″ ties, the end plates *c′*, and the wall plates *d*, belong to the sinking set. The false set was erected inside the sinking set with end plates, wall plates and the girts *g*, parallel to those of the sinking set. The false set was lined with hemlock planks *e*, to retain the concrete, and was braced at ends by the diagonals *f*. The mixture of concrete used was Portland cement, one part; sand, two and one-half parts; and broken stone, five parts. This was tamped between the outer and the inner linings up

Plan

Fig. 60

to the bunton *g*, next above, and allowed to set over night. Before putting in the next section, the buntons were removed, and struts were placed against the sheathing of the false work, thus leaving no wood embedded in the concrete.

T rails *h* were embedded to reinforce the concrete, and were anchored to the side and end plates by the combined anchor bolts and straps *i*. An enlarged view of these anchors, which were placed 6 feet apart vertically and horizontally, is given in Fig. 61. Cast-iron bunton channels *j*, with interlocking ends, were also embedded in the concrete.

These channels and buntons are distinctly shown in Fig. 59, which is the completed horizontal section of this shaft. At the point where the cribbing was forced in during sinking operations, the rails were placed vertically for a distance of 20 feet, and wire ropes 3 feet apart were placed horizontally between them.

Water was very troublesome while putting in this concrete, but was taken care of by using six 3-inch pipes having 1-inch holes in them every 3 inches. These pipes were placed in the concrete, starting at the bottom, where each had an elbow and nipple that passed through the sheathing and emptied into the shaft. It was found at first that two of these pipes at a time were sufficient to carry the water from the sides of the shaft, and, by means of troughs, the water was collected at the side where work was going on and carried into the upper ends of two of the pipes. These pipes were continued up to a short distance from the top of the shaft, and as they were no longer needed, were filled with cement grout.

Side View
FIG. 58

74. Brick Shaft Linings.—In Europe, brick masonry is used quite extensively for shaft linings; in fact, the shafts are made circular or oval in horizontal section in order to use brick for lining. Brick masonry in rectangular shafts

would resist but little pressure, but where the sides of the excavation are arched, as shown in Figs. 62 and 63, the brickwork will offer considerable resistance to pressure. It will be noticed in Fig. 62 that only 6 inches of the area on

Fig. 60

each side of the cage is wasted. This form is being adopted for concrete linings in Pennsylvania coal mines. The form shown in Fig. 63 furnishes a larger airway than that shown in Fig. 62, or can, as shown, furnish two small compartments and be amply strong to resist pressure by its arched sides. Fig. 64 represents a circular shaft lined with brick. The curb *C* on which the brickwork rests is made of wooden or cast-iron segments. The method of laying the brick is to cut a ledge in the solid rock *B*, and then build the lining to the surface *A*. When the lower section of brickwork approaches a curb above, the rock *D* is gradually cut away until the brickwork is built close to the curb.

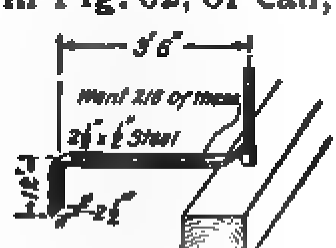


Fig. 61

75. Shaft Scaffolding.—Since sinking is carried on during the operation of walling, a scaffolding on which the masons stand is hung in the shaft. Fig. 65 illustrates the scaffold used at a colliery in the British Isles. The

Fig. 62

Fig. 63

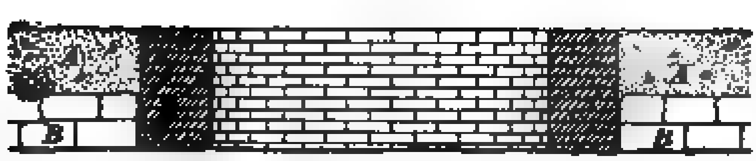


Fig. 64

platform *a* is suspended from the top of the shaft by the wire ropes *b*, and consists of steel angle bars and a plank floor. The angle bars are 4 in. × 4 in. × $\frac{1}{8}$ in. in size, are curved to the diameter of the shaft, and rest on 5″ × 5″ × $\frac{5}{8}$″ steel angle bars, which are virtually the floorbeams. Upright angle bars *c* are bolted to the floorbeams, and braces *d* act as stiffeners, being bolted at one end to the floorbeams and at the other end to the uprights *c*. The platform is surrounded by a sheet-iron railing, to prevent bricks or tools from being accidently knocked off the platform on to the heads of the men at work below, and a hood *e* is also placed above the platform, to protect the masons from falling objects. The scaffolding is suspended by means of ropes, which pass half way around the pulley wheels *f* and are wound on the drums of crab winches at the surface, thus affording a means for raising and lowering the platform.

FIG. 66

There is an opening *g* in the center of the platform through which the rock bucket *h* and the water barrel *i* pass, leaving the rider *j* at the hood. The mortar and bricks are lowered to the masons by the bucket *k*. It will be noticed that the curb *l* is constructed so as to form a combined curb and water ring, thus keeping the water from splashing on the sinkers and directing it to the water-barrel tank.

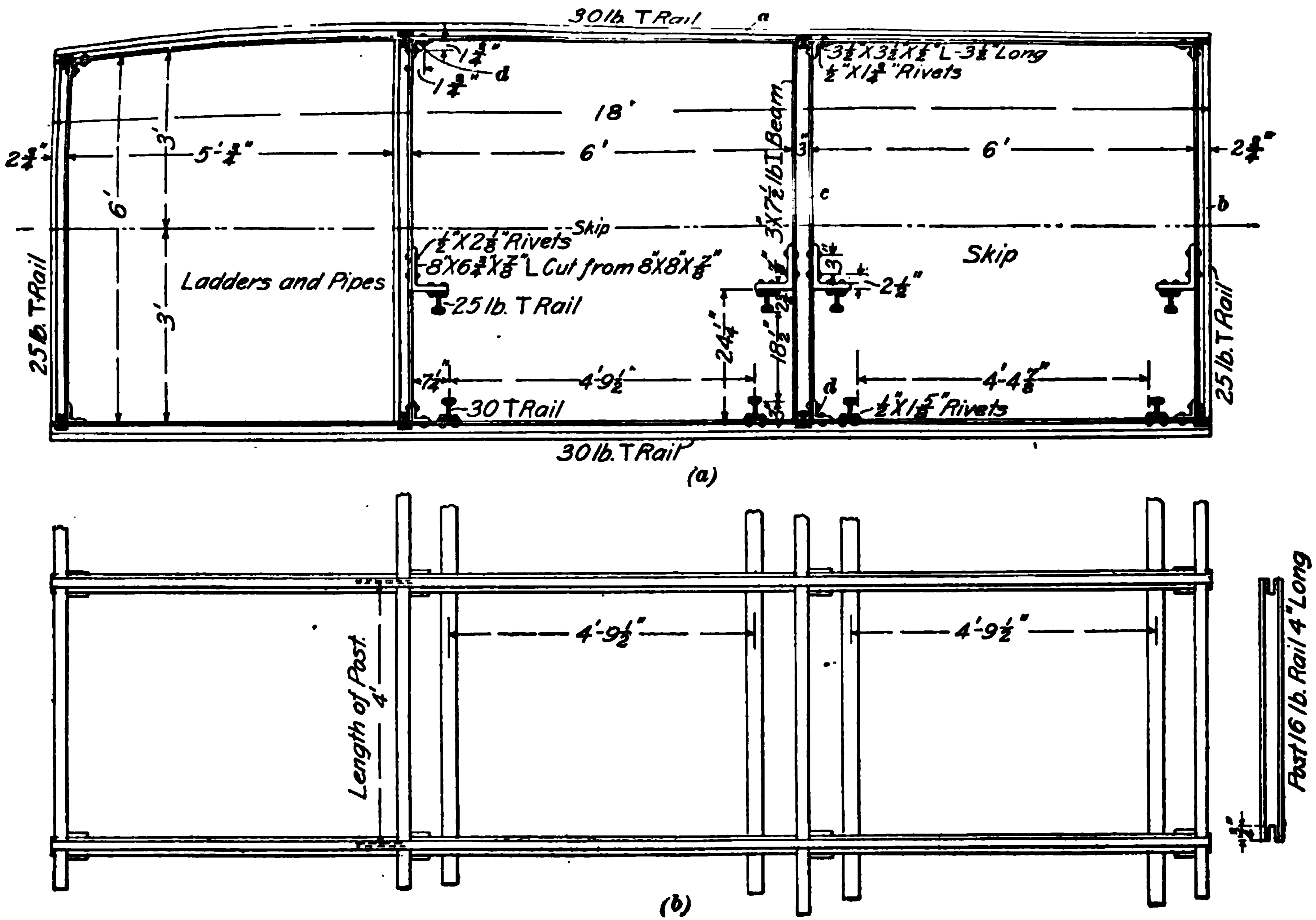

30 lb. T Rail
a
$3\frac{1}{2}$" X $3\frac{1}{2}$" X $\frac{1}{2}$" L - $3\frac{1}{2}$" Long
$\frac{1}{2}$" X $1\frac{3}{4}$" Rivets
$1\frac{3}{4}$"
d
18'
$5' \frac{1}{4}$"
6'
6'
$2\frac{3}{4}$"
3'
3'
6'
3 X $7\frac{1}{2}$ lb I Beam
c
b
Skip
Skip
Ladders and Pipes
$\frac{1}{2}$" X $2\frac{1}{8}$" Rivets
8 X $6\frac{3}{4}$ X $\frac{7}{8}$" L Cut from 8" X 8" X $\frac{7}{8}$"
25 lb. T Rail
25 lb. T Rail
25 lb. T Rail
$2\frac{1}{2}$"
$24\frac{1}{4}$"
$18\frac{1}{2}$"
$7\frac{1}{4}$"
$4'9\frac{1}{2}$"
$4'4\frac{7}{8}$"
30 T Rail
$\frac{1}{2}$" X $1\frac{5}{8}$" Rivets
30 lb. T Rail
(a)
Length of Post 4'
$4'9\frac{1}{2}$"
$4'9\frac{1}{2}$"
Post 16 lb. Rail 4" Long
(b)

FIG. 66

METAL SHAFT LININGS

76. Steel Shaft Lining.—The objection to iron or steel supports in most shafts is that they are easily corroded by acid water. A shaft in one of the mines near Ely, Minnesota, in which the water is not acid, is lined with steel. The frames used are rectangular, as shown in Fig. 66 (*a*) and (*b*), and have side plates *a* and end plates *b* that are made of T rails, those for the former weighing 30 pounds and for the latter 25 pounds to the yard. The buntons *c* that divide the shaft into three compartments are 3-inch I beams, and weigh 7½ pounds per foot. These buntons are riveted at their ends to angle irons *d*, and the latter to the side plates.

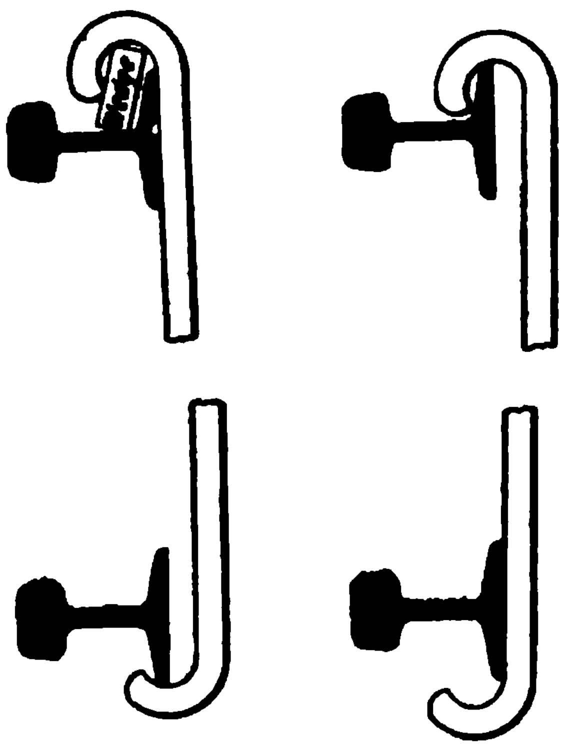


FIG. 67

The sets are suspended in the shaft by hangers, which are pieces of iron bent into the form of a hook at each end, as shown in Fig. 67. The upper end is hooked over the plates of the set above, while the lower hook holds the plates below in place. The weight of the lining is taken up at intervals by cutting hitches in the rock and inserting rail bearers.

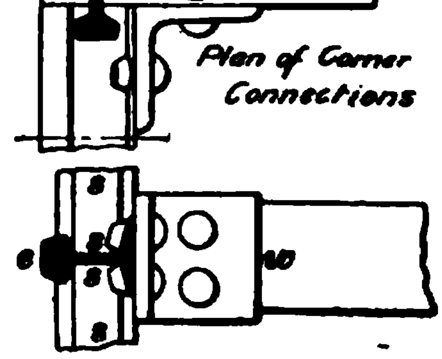


FIG. 68

The methods of joining the corners of the sets is shown in Fig. 68, where *w* is the wall plate; *e*, the end plate; and *s*, the studdle. The end and wall plates are riveted to an angle iron, while the web of the studdle is slotted at each end to receive the flanges of the wall plates, as shown in Fig. 67. No lagging is used except where the strata demand it, and then old wire ropes woven with wooden laths or corrugated sheet iron are employed. It is probable that such shaft supports embedded in concrete could be employed to advantage

in places that have acid water, for the cement would protect the supports and the supports would stiffen the concrete.

77. Iron Tubbing.—Where the pressure of water is great, and long lengths of brick linings would not be able to support the pressure, cast-iron segments that will form rings when joined together are employed in Great Britain. At one time, solid rings were used, but the difficulty of getting them into position and their liability to break caused them to be abandoned for segments. The flanges of the segments are placed away from the center of the pit in England, and toward the center of the pit on the Continent. In England, it was found that flanges bolted together could not be depended on; hence, the flanges were turned around and wooden wedges used to fasten the segments in place.

78. Cast-Iron Water Curbs.—To place tubbing, it is first necessary to make an even foundation for the water

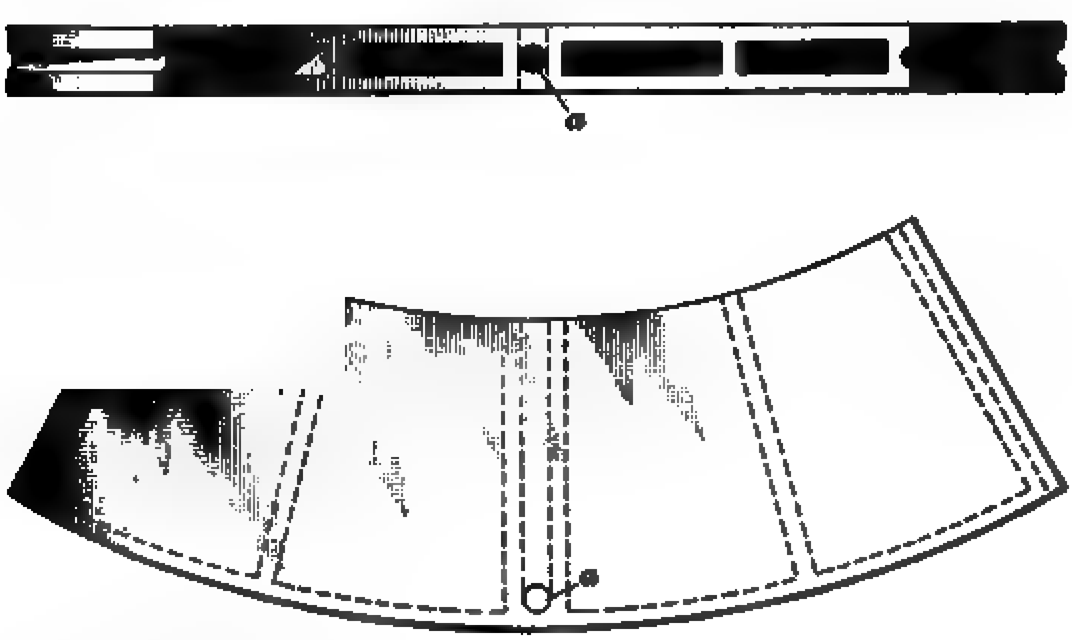


FIG. 69

curb to rest on. Formerly, the practice was to use oak curbs, but they have been abandoned for cast-iron curbs 18 inches wide by 6 inches deep, and made in segments, as shown in Fig. 69. The segments are set in position on a bed of cement, and $\frac{1}{2}$-inch pieces of wood are placed between the joints in such a manner that the end of the grain of the wood is presented to the inner part of the shaft where

wedging takes place. All around the curb, in the space between it and the sides of the shaft, is placed seasoned timber, free from knots, with the grain pointing upwards. As many well-seasoned finely tapered pitch-pine wedges as possible are then driven in between these sticks, care being taken to drive these wedges at various points, so as to evenly distribute the pressure and prevent any displacement of the segments. When no more wedges can be driven, chisels are used to make spaces, and wedges are then driven into these spaces. After one curb has been put in place, it is usually followed by another, and then a third. It will be noticed that there is a hole *a* in the curb into which a cast-iron pipe is screwed, and that this hole is carried upwards back of the lining for the purpose of draining off water and relieving the pressure.

79. Tubbing Plates.—Tubbing plates, as shown in Fig. 70, are cast in sections 2 feet high and about 4 feet

FIG. 70

long. Each segment is strengthened by ribs that meet in the center and by brackets at the edges, and each plate has a hole *b* in its center, to allow water to pass through while the operation of laying the plates is proceeding. The spaces *a* are formed by the flanges *c* on the horizontal joints and the flanges *d* on the vertical joints, and in them are placed dried pieces of wood having the ends of the grain

pointing toward the center of the shaft. In Fig. 70 (*a*) is a vertical section and (*b*) is a perspective view of a segment.

80. Wedging Tubbing Rings.—The first layer of tubbing plates is placed on the curb, and the vertical and horizontal spaces are filled with the wood already mentioned. Wedges are next driven down between the rock and the segments, so as to prevent the latter from moving. A second layer of segments is then laid on the first, and the process repeated as far up as the lining is to be carried. As the tubbing is added section by section, the space between the sides of the excavation and the tubbing is filled in with concrete or soil packing. All the horizontal and vertical joints are next wedged by driving wedges into the wood between them, using a chisel to open a place for the wedge to enter. When no more wedges can be driven in, and the holes *b* are plugged, the tubbing will be quite dry and the water will be cased off.

81. Calculating the Strength of Tubbing.—Galloway gives the following formula for calculating the thickness of a lining required to resist the pressure of water on the circumference of a round shaft.

Let T = thickness of lining required, in inches;
D = internal diameter of the shaft, in inches;
H = head of water, in inches;
W = weight of a cubic inch of water .036 lb.;
R = $33\frac{1}{3}$ per cent. of the coefficient of resistance to crushing per square inch of the substance employed.

Then,
$$T = \frac{WHD}{2(R + WH)}$$

The coefficients of resistance to crushing must be ascertained by actual experiment with the material to be employed, but the following, in pounds per square inch, are approximate: Cast iron, 80,000; brick, 1,000; cement, 3,000 to 5,000.

EXAMPLE.—What thickness of cast-iron tubbing will be required for a shaft 15 feet in diameter and having a 600-foot head of water?

SOLUTION.—

$$T = \frac{.036 \times 7{,}200 \times 180}{2\,(26{,}400 + .036 \times 7{,}200)} = \frac{46{,}656}{53{,}318} = .87. \quad \text{Ans.}$$

82. Deepening Shafts.—When a shaft is to be deepened while the upper part is in use, it may be done by leaving a piece of solid ground above the miners as a protection. If this protection consisted of wooden beams, it would be called a *pentice*, hence the term **pentice shaft**. Fig. 71 shows a piece of ground *a* left under the hoisting

FIG. 71

compartment *b* of a shaft that is being deepened. The excavation *c* is supplied with a bucket, which may be hoisted by an engine located on the level *d* or at the surface. Engines for such locations, if placed underground, should be run by air or electricity. In small shafts, a crab winch turned by men will generally answer for raising the broken material. After the required depth has been reached and new levels are started, the pentice is cut away.

A method of deepening shafts, similar to the one shown in Fig. 72, is to sink an incline *a* some distance away from the shaft, continuing until a position directly under the shaft is reached. A carriage *b* is then arranged to run on this incline between *c* and *d*, and to stop when it reaches *d*. The sinking rope and bucket is carried by the carriage, and when it reaches the stop *d*, the bucket goes into the shaft *e*. The sinking rope *f* is worked by an engine at the surface, and in hoisting from shaft *e*, raises the bucket to the carriage. The carriage and all then moves up the slope until the level is reached, when the bucket is detached and carried to the hoisting shaft. An empty bucket is then attached to the carriage and lowered into the shaft as before.

FIG. 72

83. Ladders.—While ladders are not used in deep mines, as already stated, nevertheless they are frequently employed in small or shallow ore mines. The objections to ladders arise from the difficulties that the landing platforms offer to fire-fighting, and to the fact that, in deep mines, the men become exhausted in climbing them.

Whenever ladders must be used, care should be taken to see that they are strong and well put together. The ladders, if possible, should not be placed vertical, but should be given an angle not less than 45° nor greater than 70°. The mining laws in some states provide that such ladders shall not have an inclination steeper than 60°, and that proper

landings shall be placed at the top and bottom of each ladder. Fig. 73 (*a*) and (*b*) shows two methods of arranging ladders in shafts. The method shown at (*a*) is the better, as all the ladders are arranged in one direction and there is less

Fig. 73

danger of falling than with the arrangement of the ladders at (*b*). Particular attention must be paid to the rounds of ladders, to see that they are kept in repair.

ORE MINING

EXPLOITATION

INTRODUCTION

1. Systems of Exploitation.—The operation of taking ore out of the ground is termed **exploitation**; the work of exploitation is mining. To arrange a mine so that mining may be carried on systematically is termed *development.* There are two ways of exploiting mineral deposits: one is by open work, where the mining is done in the open air; the other is where the mining is done under rock cover. These systems are modified almost continually by the introduction of machines or methods to assist in mining. Owing to mineral deposits being in the form of beds, or veins, that are horizontal, inclined, or vertical, and which may have strong or weak enclosing walls, it is necessary to vary each system of mining to meet these requirements. The different varieties of deposits are what bring into prominence the ingenuity, experience, and education of the mining engineer, and by which his ability is determined.

Probably no industry offers so many varying problems or embraces so wide a field of general knowledge as mining. Each mine is a separate problem, requiring a different solution even from an adjoining mine in the same field, but only in particulars, so that exploitation is narrowed down to where the educated mining engineer abreast of the times can cope with any problem arising.

Competition makes it necessary for the successful miner to know the latest practice in exploitation and mining methods, for which reason it is not necessary for him to have had experience in every class of deposit, since he can adjust his method to suit the deposit.

2. Duty of the Mining Engineer.—Handling material, working men, knowing how to care for machinery, and how to apply the cheapest and best method of mining form the duty of the mining engineer. This is no light matter when the property under exploitation is in a rough, precipitous, mountainous locality, as is generally the case in precious-metal mining. The property may be difficult to reach, and remote from adequate supplies of mining materials and duplicate parts of mining equipment; therefore, the mining engineer must keep in reserve, on the premises, all necessities, and he must be on the alert to meet emergencies that may arise. This will require an education that will enable him to invent and adopt means to meet varying conditions promptly when they occur; and to do so intelligently and skilfully, he must keep himself thoroughly informed, as far as practicable in advance of actual work, as to the physical features and changing conditions of the ground and mineral. This often demands an examination of the mineral exposed by each day's labor. Recent practice almost compels the mining engineer to know something of metallurgy, particularly when working large precious-metal mines, where ore is graded into smelting, amalgamating, lixiviating, and concentrating products, which are treated on the premises. Probably, the most difficult mining proposition for an engineer to work is one that employs about forty men, since in such positions the engineer must be mine and mill superintendent and assayer, unless the mine is a bonanza. At large mines, he will have assistants and can then give his entire attention to executive measures for economical working.

OPEN WORK

3. Kinds of Open-Work Mines.—Where a deposit stands nearly vertical or highly inclined, as in Fig. 1, mining may be carried on from the surface downwards, until the hanging wall *a* overlaps the deposit and threatens to fall into the excavation. This will occur when the weight of the projecting rock exceeds its tenacity and will vary for each kind of rock; hence, if there are any signs of the hanging wall weakening, the wall must be shot down or mining carried on underground. The depth to which such excavating is limited may be water level; that is, the point where water does not flow away naturally. Where a deposit is only slightly inclined, as in Fig. 2, that portion between *a* and *b* can be mined by open work, but as soon as it goes under the cover *c* one of two methods must be adopted. In case of a loose gravel or dirt cover, this may be removed, say to *d*, and deposited in the excavation between *a* and *b*, but in case the cover is hard rock, underground work should be substituted for stripping. Where the ore deposit is covered with a layer of gravel that is not too deep, the gravel may sometimes be removed and the entire mass

Fig. 1

Fig. 2

worked by daylight. The removal of the earth is termed **stripping.**

4. Working Outcrops.—In many instances, the outcrops of deposits have been attacked and mined regardless

Fig. 3

of subsequent work. This is wrong from an economical standpoint, as it leads to expensive working in the future. Fig. 3 is an illustration of a magnetic ore mine in New

Jersey whose outcrop was removed because it was cheap mining. When winter arrived, mining ceased; and after the snow and ice had melted and the water was pumped out in the spring, mining was resumed. Although the hanging walls were strong, the limit at which they would stand without support was soon reached and pillars were left at *a* and *b* to keep up the roof, thus forming a series of pits that collected water and had to be drained separately. When this method of working became difficult, so that drainage became expensive, and rainy days stopped work, a shaft *c* was sunk and underground mining carried on. The present mining engineer has had to timber and lag over the open cut to prevent the cold of winter affecting the men and freezing things in the mine generally, also to prevent snow and ice sliding into the mine and to obtain a regular system of ventilation. While there was, possibly, some excuse for open-cut workings of this character in the early days of canal transportation, there is no excuse now, even if a person lacks capital. The outcrop of a deposit of this description should never be excavated, since surface water will flow into the mines; and that, together with the natural influx of underground water, will, at times, cause trouble and probably drown out the mine.

5. Exceptional Outcrop Workings.—Fig. 4 is a cross-section of the Caledonia gold mine in the Black Hills of South Dakota. It is a low-grade body of ore *a* divided by a dike of slate *b* and porphyry *c*. The discovery of the vein was at *d* and was worked as an open cut until the slate *b* fell into the excavation. Open-cut mining was then carried on at *e* until the dangerous hanging wall compelled its abandonment and then a cross-cut tunnel *f* was driven to cut the veins. Regular stoping was then carried on, in both veins, to daylight, as shown. All surface water that came into the mine was caught and carried out through the level *f*; and because of the large openings, it was possible to use a hoisting engine to develop the mineral below and hoist it to the level *f*. The water that collected on the lower levels

FIG. 4

was pumped to the adit level *f*. The mass of ore *a* became too heavy for the timbering and eventually fell, wrecking the timber sets.

There is danger in the removal of a mass of rock that stands on a base that is broader than its upper portion, or apex, like the letter **A**; while, on the other hand, a **V**-shaped mass is largely supported by the walls. Many ore bodies are lens-shaped; that is, broader at the center than above or below, and in these masses the greater expense and danger attend the removal of the upper portion.

6. Surface Water.—When the outcrop of a vein has been worked by open cut, a series of ditches should be excavated about the opening to prevent, as far as possible, the surface water entering the mine. It is never policy to work a mine by open cut below the natural drainage level when the mining must eventually be carried on underground. If the vein material is of a porous nature, the surface water will seep into the mine, whether the open cut is above or below the natural drainage level, but particularly in the latter case. This requires continual expense for pumping, which increases as the depth of the mine increases, since more power is required to hoist water from a deep pit than from a shallow one.

The proper method to pursue is to sink a shaft and commence a regular system of underground mining; the profits will not be so great at first but will increase as the work advances; besides, this method is in the nature of prospecting and gives an idea of the value of a deposit, and the ore that may be depended on.

7. Open-Cut Mining.—The Tilly Foster mine in New York State was worked as an open cut until the hanging wall threatened the destruction of life and property by frequent falls of rock; the hanging wall also exerted such great pressure on the underground workings that it was found impossible to recover much of the deposit. To facilitate mining, a portion of the hanging wall was blasted down and hoisted out; the mine was then continued as an open pit.

Fig. 5 illustrates the method in which both the rock blasted from the hanging wall and the ore were removed. Cable tramways were arranged so as to lift the bodies of the ore cars from their trucks and lower them into the mine. When

Fig. 5

filled, they were hoisted, replaced on the trucks, and drawn to the dump.

In localities where ore deposits are in large basin-shaped masses surrounded by country rock, cuts or tunnels are made through which mine cars or even railway cars are run direct

to the face of ore. Fig. 6 shows the cross-section of such a deposit in Virginia, in which *a* is the ore body and *b* the country rock. The first attack was made through the cut *c* and, when the stope of ore *d* was removed, another cut *e* was

FIG. 6

made. This required that the country rock should be quarried out in order to bring the ore out on grade from stope *i*. The next stope required a tunnel *m* to reach the ore and bring it out on grade from the stope *n*. It will be seen that as the ore is worked out, the tunnels become longer and longer and that their cost will detract from the profits to be derived from the ore, and hence there is a limit to their

FIG. 7

length. Fig. 7 shows the interior of such a mine, where the material is broken from the bank and loaded directly into the cars. The drainage of such mines is not difficult, and during wet weather the miners work in rooms made in the ore bank. A system somewhat similar to this is carried on in some of

the iron-ore mines of Lake Superior, also in the phosphate mines of Georgia; but in the latter cases, stripping was necessary before the ore was attacked.

8. Stripping by Hand.—In some sections, where the deposit lies in nearly horizontal beds and the land is more valuable for agricultural purposes than for the ore, it may be worked, as shown in Fig. 8, by *stripping*. The soil *a* is thrown back so as to uncover the hard pan *b*. The latter is then loaded into wheelbarrows, removed to the pile *c* and dumped until sufficient of the ore deposit *d* has been uncovered to take out a good-sized slice by blasting. The soil *a* is next loaded into the wheelbarrows and spread over

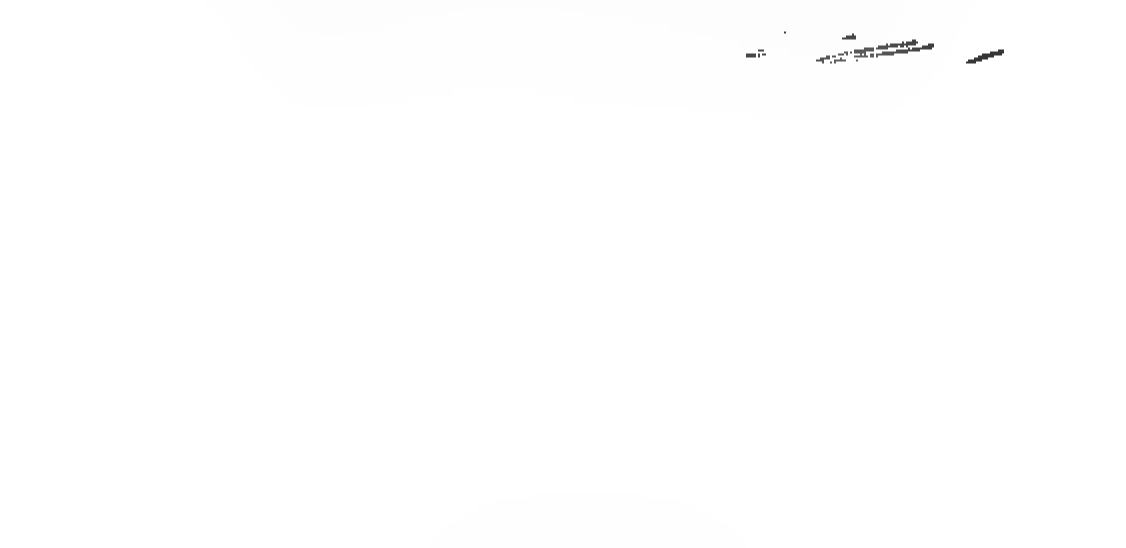


Fig. 8

the hard pan *c*, thus leaving the ground with the ore removed in practically the same condition in which it was before mining.

In order to carry on this system of mining, the ore deposits must be worth sufficient per ton to pay for handling the overburden. With labor at $1 per day, the cost of picking, shoveling, wheeling, dumping, and spreading material by hand in the United States is 25 cents per cubic yard. With carts or cars this may be reduced to 18 or 20 cents. Taking this as a basis with any other price for labor, the cost may be determined.

Example.—An ore deposit worth $2.50 per ton has an overburden of 6 feet of hard-pan soil; when a cubic yard of the ore weighs

2.5 tons, how much profit will there be in mining, with labor for stripping at \$1.50 per day, and the cost of putting the ore on the cars 60 cents per ton?

SOLUTION.—When the cost of labor is \$1, the cost of stripping $6 \times 9 = 54$ cu. ft. or 2 cu. yd. is 50 ct.; hence, when labor is \$1.50 per day, the cost of stripping is 75 ct. A cubic yard of ore is $3 \times 3 \times 3$ ft. and weighs about 2.5 T.; hence, the cost of stripping for 1 T. will be 30 ct., and the total cost of the ore on the cars is $60 + 30 = 90$ ct. per T. $\$2.50 - .90 = \$1.60$ per T. profit. Ans.

9. Stripping With Steam Shovels.—In some instances, ore deposits are stripped of their overburden by steam shovels. The usual method in such instances is to lay tracks for the steam shovel to move on and then other tracks alongside for cars that the shovel loads with dirt. The loaded cars are then hauled away and dumped, empty cars being substituted so that little time may be lost. The steam shovel makes a cut nearly the length of its boom on each side of its track and this for large steam shovels with 26 feet booms is about 50 feet. The largest steam-shovel buckets have a capacity of 3½ cubic yards, and are listed to scoop up between 1,500 and 4,000 cubic yards in 10 hours. This highest duty, however, is seldom realized on account of stoppages when changing cars; and the largest day's work on record probably is 3,200 cubic yards. Fig. 9 shows a steam shovel at work at the Oliver iron mine in the Mesabi Range in the northern part of Minnesota.

The main objection to the use of steam shovels is that they take out a slice of ore about 18 feet in height, and when work is begun on the next slice the track approach must be lowered that much, making the cuts or approaches, which are from 500 to 1,000 feet in length, longer and more expensive each time.

The work of the steam shovel can be increased by first shaking the ore banks with powder, for which purpose holes about 15 feet deep, 15 feet back from the face, and 15 feet apart are put down by drills and hammers. After this is done, a charge of dynamite is shot in the hole to form a cavity; sometimes this dynamite shooting is repeated, after which the holes are charged with black powder and fired.

Fig. [illegible]

The holes are put down parallel to the face to keep that nearly perpendicular. The cost of explosives for shaking the ground is placed at from 3 to 5 cents per ton of ore loosened. The cost of putting ore on the cars by steam shovel at the Oliver mine is given at 30 cents per ton. The cost of stripping with a steam shovel varies from 20 to 50 cents per cubic yard, the latter being the price when the hard pan is frozen and the former when conditions are favorable. It will be noticed that hand stripping compares favorably in cost with steam-shovel stripping, but the latter is much quicker as a rule, and for that reason, where a large output or quick work is desirable, it is preferable.

FIG. 10

10. Steam-Shovel Mining.—When steam shovels are used as soon as the cover has been removed, the ore can be attacked and loaded directly on to broad-gauge cars. The output can be increased by additional steam shovels, and when a mine sends out 920,000 tons of ore per year of 230 working days, as did the Oliver mine, it is at the rate of 4,000 tons daily. Three good machines will do this, and work rainy days, where, on the other hand, it would take 200 men with pick and shovel to handle the same material, and break it down.

Fig. 10 shows a steam-shovel mine with high benches for the shovel to work against. This is not desirable and

experience has demonstrated that 18 feet is the most suitable height, for above that the ore rolling down the bank against the car gives trouble, and slides that will almost cover up the machine may occur.

The cost of a good steam shovel is about $8,000, but whether it is policy to use this method of mining even in

Fig. 11

such large deposits of ore as are found in the Lake Superior iron-ore regions is debatable, some even claiming that when the dead work of stripping and all other items are considered, including water, it is cheaper and more satisfactory in the long run to work by underground methods.

11. Phosphate Mining.—Phosphate deposits in South Carolina, Georgia, and Florida are usually below alluvial

deposits, and require to be stripped. There are some, however, especially in Florida, that are below water level, and are mined by traction dredges, or by floating dredges that dig their own canals. Fig. 11 shows a back-action traction dredge. The stripping has been done by an ordinary steam shovel, and the traction dredge is working from that bench. The material is loaded into cars, which are hauled to a washer and dumped, but at some plants the phosphate pebbles are washed free from dirt at the mine. In such cases, the shovel delivers the material to a hopper and the refuse flows back into the cut made. This latter method is advisable, since it is not economical to haul worthless material and then have to dispose of it afterwards. The last remark applies to every class of mining; viz., that the place to sort ore is where it is excavated, even in precious-metal mining where high values prevail and expert sorting must be practiced in daylight. The sorting should be done at the mine, and not at some distant point.

EXPLOITATION BY MILLING

12. Introduction.—The term *mill* in mining may be defined as an auxiliary shaft sunk or raised within the mine (most often vertical, but, if conditions require, at a slight inclination from the perpendicular) to connect a working place above with a level below. In precious-metal mining, it very conveniently and economically expedites underground exploiting, not only as a mill hole for passing excavated material into mine cars, but as a temporary storage bin for ore or barren rock until other work in progress permits its removal. Such openings are often supplied with ladders to shorten the distance between the working places, and thus become manways, up and down which men travel. Mill holes offer such advantages for loading ore into cars that it has been found desirable, in cases, to place them at convenient distances (60 feet apart). When supplied with trap doors, mill holes can be made to regulate the mine ventilation.

Milling is mining ore and passing it down a mill into cars. Preliminary to such work, stripping may be required

if the deposits are of large area, like the Mesabi iron ore, or railroad cuts and tunnels may be needed when ore is to be loaded directly into the cars. By another method, shafts may be sunk at the edges of the deposit in the country rock, and drifts excavated in the ore from which risers are driven to the surface, or to the level above. The opportunities for combining other methods of mining with milling are numerous and it is not unusual for the mining engineer to use them to advantage.

FIG. 12

13. The Sellwood Milling System.—The **Sellwood milling system** is carried out by driving into the ore a tunnel, large enough to admit a 20-ton railroad car. At distances equal to two car lengths apart, raises are put up to the bottom of the stripped pit, and the ore milled into the cars with but one shift of the train one-half a car length. This method of mining is fast and cheap; in one instance, a train of twenty cars was put in, loaded, and pulled out in less than 1 hour. Allowing 15 minutes for putting in a new train of cars, this is mining at the rate of 3,000 tons per day; and if there were two or more tunnels the output would be

enormous for a single mine. The miners, in milling, blast and shovel the ore into the mills from which it is drawn into the cars as desired. Fig. 12 shows a deposit that has been mined by the milling system in the Lake Superior iron-ore region and is also a fair illustration of the conditions that occur in drift-gravel mining.

14. Mining With Mills.—Fig. 13 shows a cross-section through a milling mine in which *A* is the drift and *B* are the mill holes. The system by which the ore is broken down is shown by the dotted lines *C*, which form with the mills a funnel-shaped cavity. These cavities will enlarge until they meet at *D*, and then the raises *E* are to be driven as shown

FIG. 13

by the dotted lines. The ore is then passed down the raises *E* until practically all the ore above the drift has been removed; then another series of drifts is driven lower in the formation and the process repeated. This system is practiced at mines where it is considered easy to strip off the overburden, but where the dip is such that it is not desirable to handle the ore with steam shovels.

Exploiting is accomplished by sinking inclined shafts, two skip tracks wide, to a convenient depth, and then driving a level from the shaft bottom into the ore. From the level, drifts are worked right and left and raises driven from them; the latter are provided with loading aprons so that it is not necessary to handle the ore when loading mine cars. The ore is hauled from the mills, in 3-ton cars, to the shaft

bottom, where it is dumped into skips and hoisted. The ore, if broken in too large pieces, will choke the mill and cause extra labor to free it. To obviate this, sollars, or platforms, are placed around the mill so that a man can stand and sledge up the masses that are likely to choke the mill. Another method is to divide the mill into compartments by timbering so that a man standing on a ladder or platform can break up the jam with a bar. There are objections to the latter method of freeing the mill as the ore is likely to start with a rush and run into the level by knocking out the loading gate. At the Auburn mine of the Minnesota Iron Company, 3,300 tons of ore were hoisted from the mine and dumped in 10 hours where this mining was practiced. One advantage of this system is that the area stripped need not be so large as when mining with steam shovels, and hence pumping is lessened. Another advantage over steam-shovel mining is that no expensive cuts are required unless broad-gauge cars are placed in the mines. If the deposit is wide and not more than 300 feet deep, an area can be entirely worked out and the hole used for the dirt from the next strippings.

CLOSED WORK

UNDERGROUND DEVELOPMENT

15. Tunnels, Levels, Drifts, and Cross-Cuts.—To develop prospects to show that they are of value as mines, levels are run from the shaft right and left at intervals of from 60 to 100 feet measured on the dip. In case it is unadvisable to sink on the deposit, shafts are put down in the country rock and cross-cuts made to the deposit in which levels are driven, or levels may be made parallel to the deposit in the country rock and cross-cuts driven from the levels at intervals to the deposit. In some cases, in fact wherever it is possible without great expense, adit levels are driven on the ore, or cross-cut tunnels in country rock, to tap the ore above water level. These openings are made

not alone for moving ore to the shaft or to the surface, but are for drainage, ventilation, general development, and exploration. They should be driven in as direct and straight a course as practicable, and in any case should avoid abrupt curves. They should also be driven at uniform grades not to exceed 7 inches rise in 100 feet horizontal measurement, nor have less than 4 inches rise in the same distance; at the outside mouth of the tunnel or in the yard, the rails may be given a grade of 1½ inches in 100 feet to facilitate the stoppage and handling of the mine cars.

16. Ditches.—At the opening of a tunnel and, later, of the cross-cuts, there may be no evidence of water, yet as the excavation penetrates the mountain, water seepage and water courses may be and usually are encountered; therefore, ditches should be provided from the commencement of the operation. The usual method of making a ditch is to blast up a channel at the side of the level, if possible on the foot-wall side; however, if the hanging wall is the wet side, the ditch may be on that side or in the center of the level under the car track. It should be made as free from obstructions as possible and kept clear of refuse, which will obstruct the flow of water.

In case the side walls of the levels are weak, they must be supported by timbers; and in some systems of mining, the levels must be timbered whether the walls are weak or not.

17. Mine Cars and Tracks.—The design, weight, and capacity of mine cars will depend on the size of levels and the character of the material mined. In iron-ore mines, the cars are made to hold from 2 to 3 tons. The wheels are made 18 inches in diameter and are shrunken on 2½- to 3-inch axles that have at least a 36-inch gauge. Such cars are usually supplied with brakes, bumpers, and couplings.

Fig. 14 shows a mine car used in the precious-metal mines of the West. The track gauge for these cars is 18 inches, the track being made with T rails weighing 12 pounds to the yard, secured at the ends by fish-plates, bolts, and nuts

and spiked to ties having 18-inch centers. While this is the customary weight of rail, it is too light for cars carrying loads above 1 ton, from the fact that the rails soon spread, and are continually in need of lining up; besides, the rails bend and are difficult to straighten. Rails weighing from 18 to 30 pounds per yard will be found much more serviceable where a large tonnage is handled.

The average diameter of car wheels at precious-metal mines of the West is about 10 inches; and while it is understood that cars supplied with larger wheels roll easier, the additional height added by large wheels is apt to make the car top heavy and liable to discharge its load. Ordinarily, car wheels are loose on the axles; and recently sand-proof

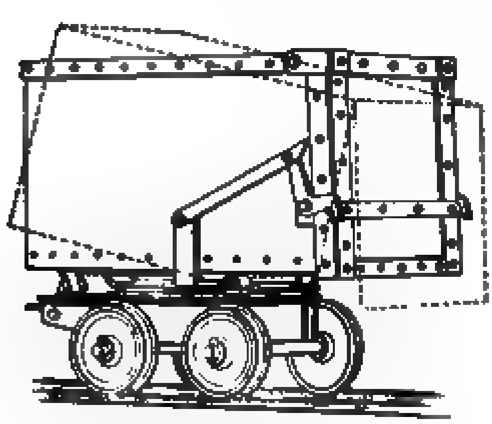


FIG. 14

grease cups have been added to the wheel hubs to cover the outer end of the axle. The practice is to fill these grease cups with a mixture of grease and graphite in order to make them self-lubricating. Self-oiling car wheels used in the East have their boxes stuffed with cotton waste, which is kept nearly saturated with cheap lubricating oil in summer and zero oil in winter. The lubricant always being soft, the cars start easily.

The conditions for a single ore car apply with the same force to a train of several cars when drawn by animals or motive power, only that in the latter case friction brakes should be on the cars in order that the driver or trammer may control them. The tracks in a tunnel should be equipped with automatic switches, or as they are known in

some localities, spring latches, so that the outgoing and ingoing cars may pass without delay, provided that double tracks are not laid the entire length of the tunnel. Fig. 15

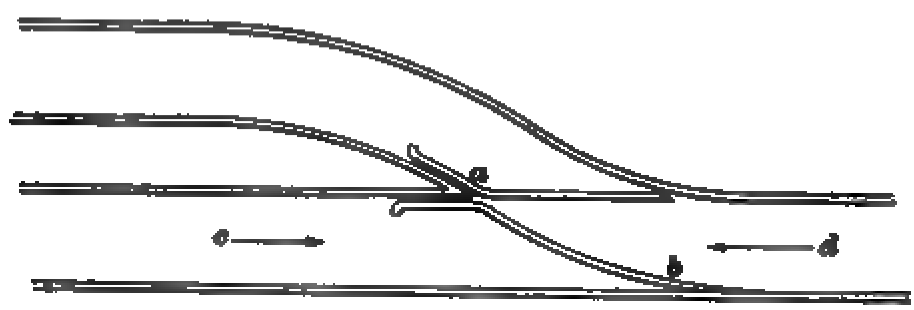


Fig. 15

shows a method of making a turnout. *a* is known as the frog; *b* as the switch point or latch, which is sometimes worked by a spring, at other times by a lever. Cars moved in the direction of the arrow *c* will open spring latch *b* automatically, but the latch immediately closes and a train of cars coming in the direction of *d* will take the curve and pass on to the siding.

18. Lateral cross-cuts, drifts, and levels should be equipped with suitable switches, turntables, or turnplates, at those points where their tracks intersect with the main working track. Sometimes, flat boiler plate is a very efficient substitute when turntables, turnplates, or switches are not available. The ball-bearing turntable shown in Fig. 16 while rather more expensive than the ordinary turnplate, has proved, generally, to be much more efficient. Fig. 17 shows a turnplate that answers in many cases where it would be impracticable to use either a turntable or a switch.

Fig. 16

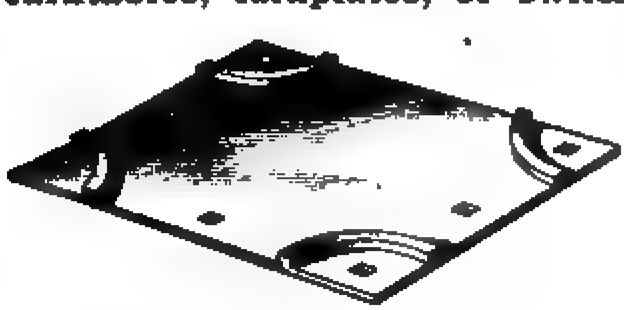


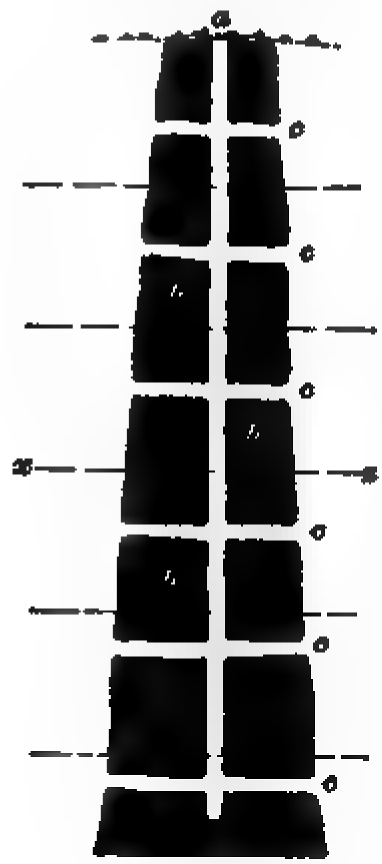
Fig. 17

19. Cross-Cut Levels.—When a vein is worked through a vertical shaft *a*, Fig. 18, **cross-cut levels** *b* and *c* are made from the shaft to the deposit, and then levels are run lengthwise of the deposit or parallel to it on one of the walls. It will be noticed that the cross-cuts become shorter with depth until the deposit is reached, when they become longer. The cost of driving cross-cuts after the vein crosses the shaft at *a* will increase with depth on pitching veins, and the question will arise whether or not it will be cheaper to sink another shaft *d* to the left of the present shaft. To decide this question, a bore hole should be put down to determine at what depth the shaft will cut the deposit, and ascertain if the vein is continuous with depth, for without a previous knowledge of these conditions, the cost of sinking the shaft may be wasted money. Another method of proving the continuance of the vein is to sink on the dip; this method, however, is not advisable from an economic standpoint, but should be done rather than sink an expensive shaft or drive a long and expensive cross-cut tunnel from the surface haphazard.

FIG. 18

20. Working on the Vein.—Fig. 19 (*a*) is a cross-section of a vein that, like most veins, is not regular in its hade, or inclination from the vertical. When shafts are sunk on such veins, the hoisting tracks must follow the foot-walls, and their difference in inclination offers every facility for the hoisting cars, or skips to leave the tracks. The hoisting rope will also be subjected to much wear by moving over the roof and floor, as well as thrashing. Fig. 19 (*b*) shows that the shaft *a* is lined on each side by pillars of mineral *b*, for the purpose of keeping the shaft open. It will be noticed that the pillars increase in size with depth. All the levels in this longitudinal section are not the same distance apart

vertically, but they are on the dip, as shown by the cross-section, Fig. 19 (*a*). Supposing that they are 60 feet apart on the dip and that there are six levels, there would only be five levels, as shown by the dotted lines *x x*, if they were 60 feet apart vertically. The levels are laid off on the dip



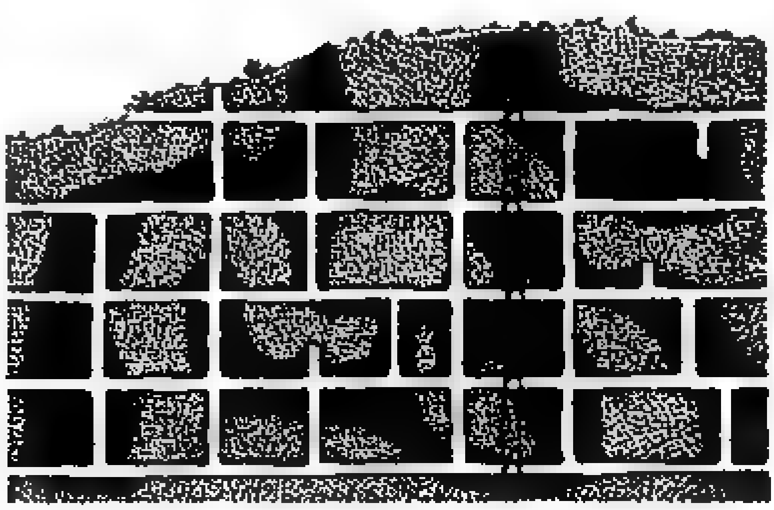
(*a*) Fig. 19 (*b*)

of a deposit for the purpose of making the ore blocks as nearly equal in size as possible, which would not be the case with pitching veins and levels measured equal distances apart vertically.

21. Blocking Out a Mine.—In precious-metal mining, the ore is not attacked and entirely worked out as the mine development progresses, for the reason that the quantity of paying ore is uncertain. It is customary, therefore, so long as there is ore obtained by development to mine only a trifle more ore than is sufficient to meet running expenses. Precious-metal veins are not, as a rule, one continual deposit of even value, such as iron-ore deposits for example, and unless the vein is in the nature of a bonanza or chimney, there will be rich patches of ore scattered through an area of

poor ground. This makes mining uncertain and may make it imperative to do dead work in order to reach a rich piece of ground, which is another reason why ore reserves, that is, the ore blocks, should be left, otherwise assessments may be required.

Fig. 20 is an ideal cross-section and longitudinal section of a precious-metal mine. It will be noticed that the ground is not blocked out in regular squares but that raises and winzes are driven in the darker places, which represent ore patches. This method of development saves money, and at the same time explores the ore to advantage; besides, the ore obtained in the work will probably more than pay the cost of driving. In the case of very rich deposits, which



(a) FIG. 20 (b)

occur in small pockets scattered through the rock, the mining must be carried on by small passages following ore stringers. This is sometimes called **coyoting** and **gophering**, but even when the mine has been worked in this irregular fashion, it is best to run levels at intervals from the shaft in order that the ground may be systematically worked and landing stations made where the cars may be brought to the shaft. In every class of mining, it is economical in the end, although returns are slow at first, to drive to the limit of the deposit and rob the mine back, rather than rob a mine for the ore in sight without regard to supports, reserves, or any future consideration. One other rule in mining that will prove valuable is to follow the ore, and in early development

work, follow it down whatever its pitch may be. Whenever the outcrop appears on the slope of a mountain, drive in a level on the vein, for it is from definite information so acquired that the data for exploitation plans is provided.

22. **Connections Between Levels.**—When a hole for an air passage or any other purpose is driven from a level to the level above, it is termed a **raise** or **upraise**; on the other hand, when an auxiliary shaft is sunk from a level to the one next below, it is termed a **winze.** Ordinarily, it is cheaper to drive a raise than to sink a winze; and where the formation is very wet, the raise has a decided advantage. It may become not only uncomfortable but even impracticable for miners to perform a full amount of labor in the top of an upraise; and, again, it may be impossible for them to work at all because of increasing heat, entire absence of ventilation, and the further fact that even one burning candle consumes too much oxygen and adds too much heat. Air for ventilation may be maintained by introducing a canvas hose, which may, in case of a good ventilating current, on the level be supplied with a cowl; but if the air-current is poor, it may be attached to a fan or some other arrangement for forcing air to the top of the upraise. A canvas hose of proper area is suitable for this work because it is flexible and may be quickly removed for blasting and as quickly replaced for driving out powder smoke. Where electricity is used in a mine, a small portable electric fan is the most convenient and efficient ventilating apparatus, but it must be especially constructed as a blower, and not be an office fan.

In the case of mines whose deposits are uniform, winzes or raises may be driven every 100 feet; but in case of irregular deposits, this practice is not advisable. In driving raises there is danger that is not noticeable in sinking winzes, namely, from loose ground falling on the miners' heads. There is trouble from water when winzes are sunk in wet mines, besides the ore has to be hoisted when broken.

23. **Cost of Driving.**—The cost of driving sinks and raises varies according to the locality and rock, but on an

average a winze 5 ft. × 6 ft. will cost $8 per running foot, while a raise will cost about $6, not including timbering. When the work is done in ore, the actual cost will be about the same, but the mineral may more than pay for the work. When drifts are less than 30 square feet in area, they are difficult to work and their excavation will cost more; on the other hand, when they are more than 30 square feet in area, they will increase in cost on account of the extra material to be broken and handled. The average cost of driving tunnels in Colorado is about $10 per foot; and that of sinking shafts, $30 per foot. The Newhouse tunnel, which is 12 ft. × 12 ft., cost $28.80 per running foot, but this is nearly three times as large as an ordinary drift and is 2,959 feet in length, making the handling of material expensive. Part of this tunnel has been driven for $20 per foot.

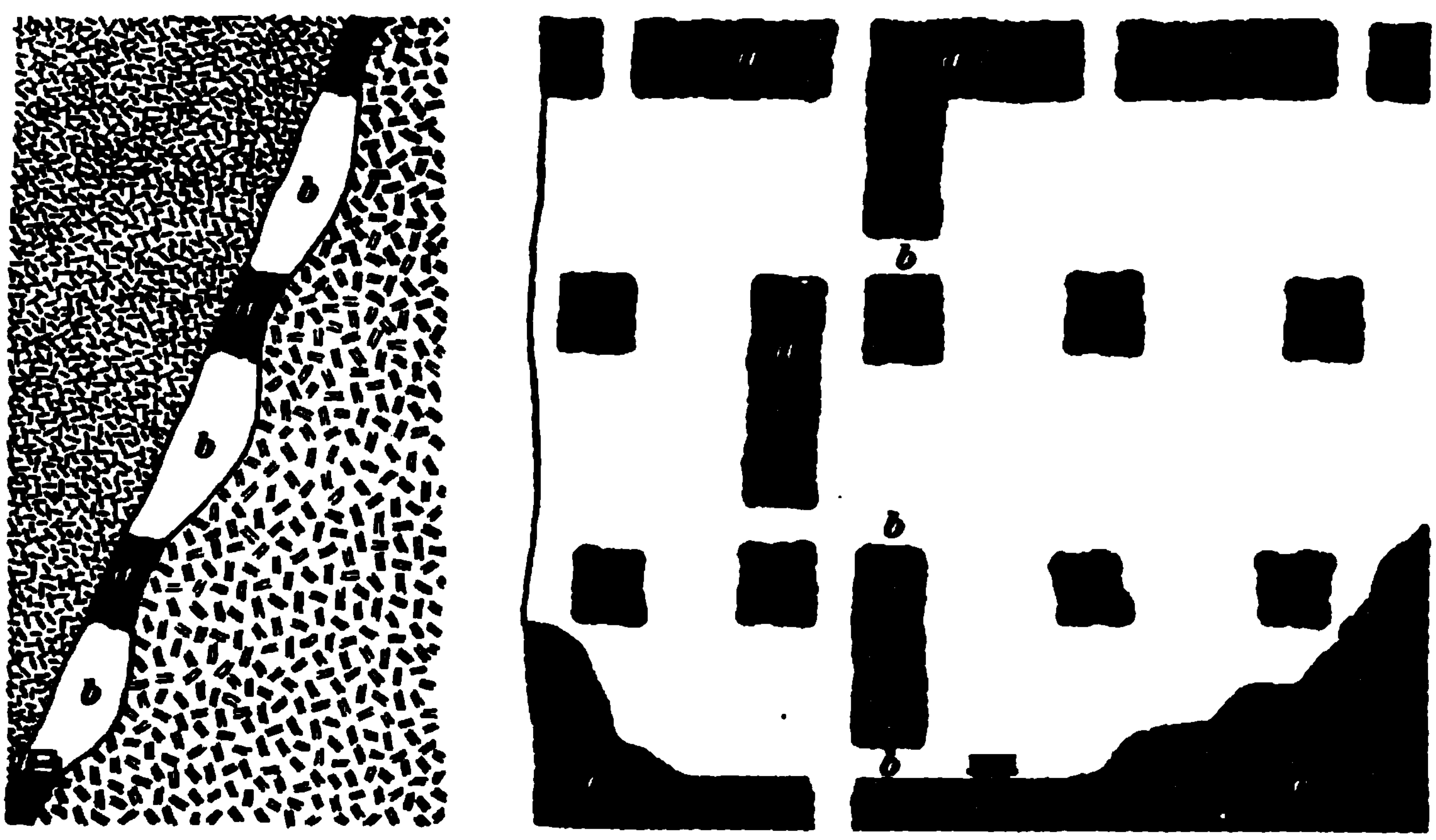
MINING SYSTEMS FOR MINERAL DEPOSITS NOT OVER 12 FEET THICK

24. Long-Wall Mining.—The **long-wall system** of mining is applicable to bedded deposits that are nearly horizontal or not highly inclined. The method of working, as shown in Fig. 21, consists in undercutting the deposit, and

Fig. 21

breaking it down. The roof must bend or break as the face *a* is advanced, in order to take the weight from the ore and the props *b*. As soon as one undercut is broken and loaded out, the back prop *c* is pulled and placed near the face. If, when the prop is pulled, the roof does not fall, it

must be shot down, for working otherwise will be dangerous. This system of working will answer for soft deposits, such as salt and clay ironstone, at least whenever they can be undercut with a pick along their face for some distance without leaving pillars of mineral. In some instances, roadways are made through the mineral, radiating from the shaft like the spokes from the hub of a wheel; in this case, the deposit is attacked between two roadways at the boundary of the property and worked long-wall toward the shaft.



(a) FIG. 22 (b)

25. Pillar and Room.—There are several systems of **pillar-and-room work**, varying, from necessity, according to the thickness of the deposit. Fig. 22 is an illustration of the system practiced at the magnetic iron-ore mines in the eastern part of the United States, where the walls are good and the ore quite tenacious. These deposits have rolling foot-walls in some parts of Orange County, New York, while the roof is quite regular, thus forming a series of lenses. Where the pinches come, a pillar of ore *a* is left and levels *b* are run both ways from the shaft until the limit of the deposit is reached. When the latter occurs, pillars about 12 ft. × 12 ft. are blocked out and the ore stoped out on the next level. Before one level has reached its limit

north and south, a sink is made and everything is in readiness for attacking the stopes *c*. It is possible to work several levels at once if the demand for this character of ore will warrant. No timbering is needed in such mines except for the shaft ladders and landings and an occasional stull.

FIG. 23

Fig. 23 shows the Clark mine in Rockland County, New York, which was worked first as an underground mine and then stripped. The rolls that occur in the foot-wall are shown at the right. The figure also shows the pillars of ore left to support the roof, which in this particular case is nearly all ore.

MINING NARROW VEINS

26. Underhand Stoping.—Stoping means to carry on work by means of steps or benches. The method of working by **underhand stopes** is shown in Fig 24. The engine shaft *a* is sunk to a moderate depth, so as to have a secure roof on the first level *b*, say from 20 to 30 feet. The first level is then driven toward an air-shaft *c* so as to secure proper ventilation. The shaft is next sunk to a depth that will furnish a stope of the proper height and a second level *d*, driven as wide as may be necessary for cars and at least

7 feet high. Between the two levels a solid body of ore is left for a shaft pillar. A raise is then commenced and directly above it, at the same distance from the shaft, a winze; these are worked until they meet. The mineral between *b* and *d* is now attacked and at the same time a row of stempels, on which rubbish is thrown, is put in above *d*. If the quantity of refuse is large and heavy, the stull timbers must be heavy and strong. At a distance, determined beforehand, another level *f* is started, and so on down the shaft. The distances between levels are divided into stopes *s* about

Fig. 24

7 feet high, so that each miner can reach the whole height of the stope he works. All the rubbish from two stopes is piled on the timbers, and serves as a road for carrying ore to the shaft and conducting fresh air to the stopes. Temporary platforms connect the stopes with the timbering sustaining the rubbish. The ore is not dumped directly down the shaft but into the break-throughs forming the shaft pillars; from these it is loaded into buckets or cars and hoisted. Other methods of underhand stoping are carried on in the ore reserves previously mentioned or in working ground at a distance from shafts.

If the work extends along the strike of a vein, the miner selects the points where the ore seems best; or, if the ore is continuously workable, he chooses points at convenient distances apart, say 150 or 200 feet, and begins to sink by cutting out a block, 6 feet or more long, to a depth of at least 6 feet. This gives the first stope floor *g*, Fig. 25 (*b*). From this floor, work is continued each way, leaving tracks and timbers, if any exist, overhead until a sufficient distance has been run to allow room for cutting out the next floor *h*. Work then goes on simultaneously on both floors each way, floor *h* being kept about 6 feet behind the upper floor *g* until room is gained on *h* to permit sinking to floor *i*, which will

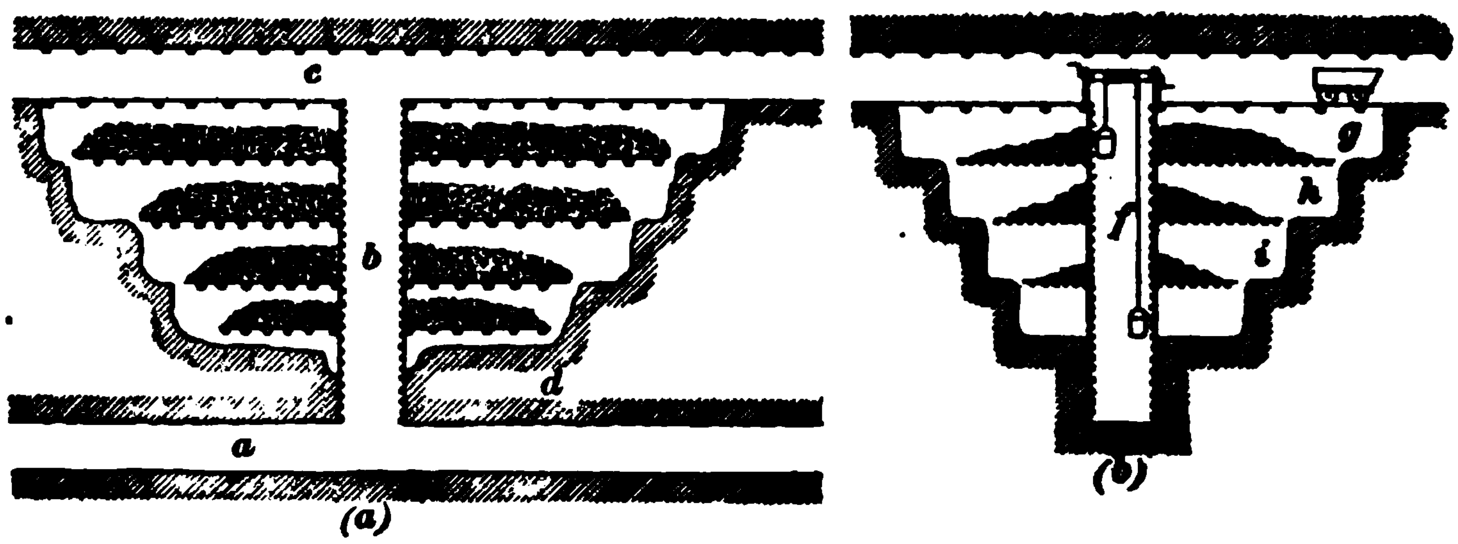


Fig. 25

have the same height, namely, 6 feet. In this way, the work proceeds until the next level below is reached or the ore gives out. If there is much waste rock, platforms are made on stempels put across the stope, on which the waste material is thrown. The platforms must be secure enough to protect the men on the lower floors. As stempels are generally necessary to support the hanging wall if the vein has any considerable dip, the expense of hoisting the waste can be avoided by using these platforms for its stowing. It may be necessary to install some form of pump to remove the water collecting in the bottom of the stope, but possibly the buckets will be sufficient for the purpose.

27. Advantages of Underhand Stoping.—The advantages of underhand stoping are that ore can probably be worked to the next lower level, hence mining and shaft

sinking go on at the same time. The work of sorting and mining ore on the stope is easily performed and the fine rich ore readily picked up; this is a matter of considerable importance when ore is friable, as then the small particles are sometimes the most valuable portions.

28. **Disadvantages of Underhand Stoping.**—Should the quantity of water be such as to require pumping or continual bailing, there will be an added cost to mining by underhand stoping. The timbering must be strong and substantial to prevent material falling on the stopes below, as the miner has no means of testing the timbering in the stopes above. The stopes cannot well be retimbered, consequently particular attention must be given to this matter or loss of life and possibly the abandonment of that portion of the mine will result.

29. **Cornish Method of Underhand Stoping.**—The **Cornish method of underhand stoping** consists in driving a level *a*, Fig. 25 (*a*), to beneath the point where the stope is to be started. Connection is then made by raising from the lower or sinking from the upper level as at *b*. This permits the material to be thrown to the level *a* and drains the stope thoroughly. The work of cutting out the floors and breasts begins at the top of the winze and proceeds as in the former case, except that the material is rolled or allowed to slide to the lower stope floor, from which it is loaded into cars on level *a* and trammed to the shaft.

The stoping described is two-winged, that is, stopes are made on each side of the shaft, but this is not always the case and one-wing stoping may be followed.

The Cornish method of underhand stoping permits greater economy in ore handling, provided that the level *a* must be driven eventually, than the former method. There is no water to hoist or pump and the ventilation is good, in fact this is a more desirable method of stoping than the former in every case except where quick returns are needed. The level may also be protected by a pillar of ore and so protect the lower stopes until it is desired to rob the pillars or

remove them. In case the lower stope becomes excessively long, a break-through to answer as a chute can be made to level *a* at any time.

The objections to the system are, that the stópes are sometimes made rather too steep to work on, and the men are apt to slide. Such stopes must be carefully cleaned from the top down so that loose pieces of rock or ore will not strike the men below. The timbering is costly and must be substantial, but even then the timbers do not last a great while in consequence of the water coming down the walls, the bad air, and the rubbish piled on them. As the future of the mine depends in a measure on these timbers, unless a pillar of ore above the level is left, it is imperative that they should be of sound, strong wood. By leaving pillars, mineral must be left, which, by some, is considered objectionable; however, with systematic mining, these pillars are in the nature of reserves and can be recovered without great expense or danger when the level is to be abandoned.

Fig. 26

30. Overhead Stoping.—In **overhead stoping**, the steps are inverted, as shown in Fig. 26. The shaft is sunk and levels started right and left at intervals. If ventilation, hoisting, and drainage are done through one shaft, it must

have compartments for the purpose, as shown; however, two shafts are always advisable in mines employing as many as forty men, since better work can be done in good air and accidents will not be so frequent. Men will slight work in places where they are extremly hot and nearly smothered, and one need not be surprised if good ore is thrown away and poor timbering done in such places. With opening of levels *a* and *b*, mining is carried on until, with the exception of shaft pillars in case the shaft is sunk on the mineral, the ore is all removed between the levels. Strong timbers or stone arches are required above each level, for if they are lost by fall of roof and material above they will be difficult to recover.

The miner commences on the level and takes out the first step above the level, standing on the ore and rubbish *c* to accomplish this. Timbers are placed back of him above the level and the next stope commenced. If the vein is all ore, about two-thirds of it may be removed when broken, the remainder forming a platform for the miner to work on. Continuing in this way, a series of stopes are worked up to the level above from both sides of the shaft. The arrows indicate the air-current that affords the men ventilation. In case the system of overhead stoping is practiced at some distance from a shaft in a mine, it will differ somewhat from that described.

31. Single Overhead Stoping.—In the general description of overhead stoping, working places each side of the shaft were represented; but it is sometimes necessary to avoid working barren ground or for some other reason to commence work on one side of a winze, as illustrated in Fig. 27. In this case, the level has been run ahead of the stopes, so that several stopes may be working on the same level *a*, and for that reason the levels must not be stopped up. The first stope *c* is advanced on the timbering and a chute *s* commenced as soon as it becomes unhandy to throw back the ore. The stopes *d*, *e*, etc. follow in order as previously described. The air travels up *s* and out into the winze *b*,

places being made in the latter for that purpose. The waste material on the stopes is necessarily full of spaces; and since the ore and refuse fall together, much fine ore will be

FIG. 27

lost unless some means are provided to prevent its falling directly on to the waste. Canvas sheets, sheet iron, or boards are therefore placed under the stope where the ore will fall, and after the refuse is removed the ore is placed in the chute.

32. Overhead Stope Carried as a Breast.—Fig. 28 illustrates a modification of overhead stoping employed in moderately pitching veins, where the roof or hanging wall

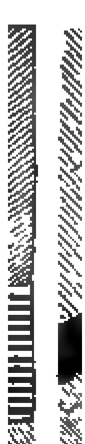


FIG. 28

requires considerable support. After the main drift *a* is driven, that portion of the vein between raises *b* and *c* is advanced as a stope; and the two raises are timbered as the work progresses. The hanging wall is kept in place

by timbering these raises and by stulls placed in the space between them. One or both raises may contain chutes *e* for the ore and one the ladderway. Waste material from mining is packed in the space *d* between the raises. In this way, the face is advanced until the next upper level is reached. After these stopes have been worked out between two levels, the pillars may be removed in the same manner, the ore being sent down through the raise used for removing it from the rooms, the roof being held in place by stulls and waste rock *d* as the ore is removed. In case ventilation is poor, a raise may be carried, in advance of the work, to the next upper level. While removing the pillars, the work is pushed with all possible rapidity; and after it is accomplished, the hanging wall is allowed to settle on the packing. When advancing by this method, the blasting is done by uppers or breast holes; and if the vein is thin, it is customary to remove a portion of the foot-wall and then blast down the ore on canvas or planking. If the foot-wall is very much harder than the hanging wall, it may be necessary to remove the hanging wall first and then blast up the ore, but this is not considered good practice. When it is necessary to fill the excavation with rock in order to support the hanging wall, it may be necessary to continue a raise in advance of the stope to the next upper level and take into the stope filling material brought from some part of the mine. This, however, need not be the case in all instances, as broken rock will occupy about 70 per cent. more space than the same rock would in the solid; and from conditions illustrated in Fig. 28, there would be more than sufficient broken material for packing. This is the case when the vein is thick and practically the entire deposit is of sufficient value to pay for its extraction.

33. Overhead Stoping From Bottom of Winze.—It is not always necessary to extend the shaft and drift under the stope before overhead stoping is commenced. Fig. 29 illustrates a case where a winze has been sunk and stoping commenced at its bottom. *a* is the winze; *b*, the upper level;

c, the drift, which, if the shaft is subsequently carried down, will become the lower level. The disadvantages of this method are that hoisting and pumping equipment, also means for ventilation, have to be provided. The advantage of this method is that the ore may be attacked by the overhead mining system and removed in advance of a level.

Fig. 29

34. Overhead Stoping Without a Winze.—Fig. 30 illustrates an overhead stope that has been carried up without a winze. In mines where the entire vein is to be removed and the deposit is practically vertical, this method is frequently employed, the men standing on platforms *c* while they work. A sufficient amount of ore, or refuse, is allowed to remain on top of the level timbers to protect them during the process of stoping. When the stope has reached the next upper level, if the material left in the stope is ore, it is drawn out and hoisted. At times, only timber in the platforms may be recovered; but the drift timber itself is sometimes drawn, provided that the entire vein is so valuable as to be removed. In one mine, this method

was continued until there was a stope over 500 feet high without a stick of timber and from which every pound of ore had been removed, but in this instance, which was unusual, the walls were very good.

35. Advantages of Overhead Stoping.—The methods of overhead stoping described are especially applicable to vertical or highly inclined veins. The method illustrated in

FIG. 30

Fig. 28 is applicable to veins having such an inclination that the ore will slide through chutes to the level *a*, and yet where the formation is not so steep but that the hanging wall exerts considerable pressure on the timbering.

Leaving out the case where overhead stoping is practiced from the bottom of a winze having no level connections, some of the advantages of overhead stoping are: No hoisting, pumping, or bailing is required in the block of ore being worked, as in the case of underhand stoping without

a winze. Water gives little trouble in the stopes. Less timber is required and much of that used may be recovered. Gravity assists in breaking down the ore. In general, it is not only more economical than underhand stoping, but leaves the stope in more workmanlike shape, and if the mine is temporarily abandoned, it will be found in better condition when work is resumed.

36. Disadvantages of Overhead Stoping.—Some of the disadvantages of overhead stoping are: There is, at times, greater danger than in underhand stoping; but as the miner is always close to the roof, he can test its firmness by tapping, and if it sounds hollow, he can knock off the insecure rock. When drilling is required for blasting, all drill holes must be uppers and this is inconvenient. In overhead stoping, there may be great loss of fine ore, which becomes mixed with the waste material, but this can be obviated by laying down canvas, sheet-iron, or board floors, for the material to fall on as it is broken.

NARROW FLAT DEPOSITS

37. Block Mining.—Flat veins having a continuous ore body require lateral drifts placed much closer to each other than the levels of an ordinary mine. This is especially necessary on account of the fact that the ore will not slide on the floor in chutes to the levels, but has to be shoveled from the place where it is broken to the car. On this account, levels are placed from 35 to 40 feet apart, as shown in Fig. 31. In this case, *S* is the shaft; *C*, the levels in the ore deposit *V*; and *E*, *F*, two cross-cut levels. The levels are connected with the cross-cut levels through the raises *D*. This system divides the ore body into a series of blocks between the levels *C*. These blocks are usually attacked from both sides—that is, from the upper level and the lower level—the miner shoveling the broken ore into a car or to a platform near the track on which the car runs. The waste material may be packed into the workings behind the miners as they advance. The two cross-cut levels *E*, *F* are assumed

in this case to be 30 feet apart. When the blocks between the levels C are attacked from both sides, it may be necessary to first drive raises between the levels for drainage,

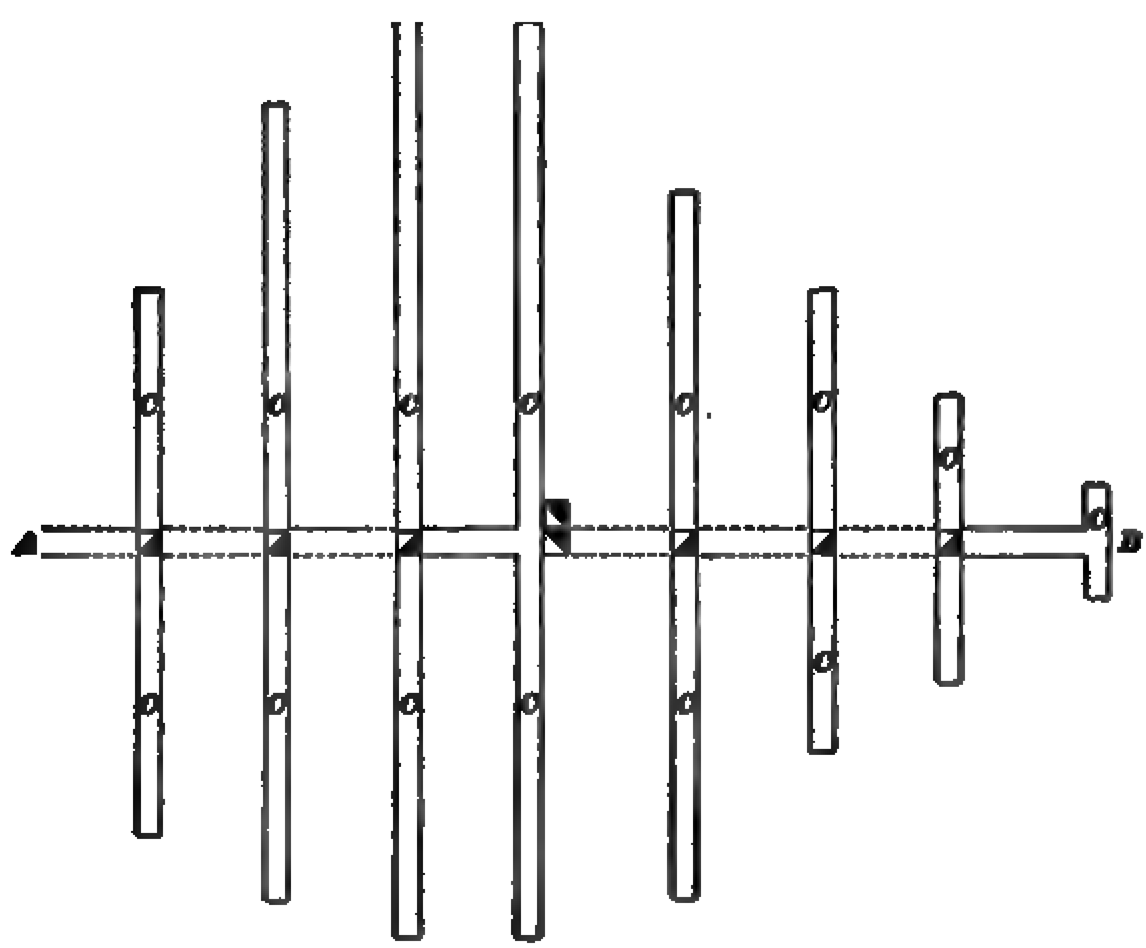


FIG. 31

and the method of attack will then be the same as in underhand and overhead stopes; that is, the advance will be by successive steps until the entire deposit is removed, the roof being supported on the waste material.

MINING WIDE ORE DEPOSITS

38. Veins More Than 8 Feet Thick.—When large masses of valuable ore occur at such a depth from the surface that stripping is out of the question, it becomes necessary to exploit them by a special system of underground work. The support of the excavation becomes so important in such instances that it forms with the ore breaking special methods of mining.

In many cases where a stope is not more than 8 feet wide, stulls are set across with plank lagging for floor, and this is all the timbering required when the walls are firm. In the case of crumbling walls that require support, square sets are necessary. In stopes of a width only requiring one length of timber to reach across, but also requiring posts, the sill stulls are set first in an overhead stope and the cap stull set first in an underhand stope. In all cases, the stulls are cut of such a length as to fit tightly against both walls, or a wedge is driven between the wall and the end of the stull to make it very tight, and each stull is so set that the upper angle that it makes with the hanging wall is slightly greater than a right angle, in order that the least settling of the hanging wall will tighten the stull. The side posts are set after the stulls are in place, in cases where the lower stull is set after the upper in underhand stopes. In overhead stopes, the posts can be set on the ends of the lower stull and the upper stull driven down on the tops of the posts. When the side posts have to be set after both top and bottom stulls, it is well to have the tops of the posts framed for gains in the ends of the stulls and drive the bottoms of the posts to place and secure them by drift bolts. The lower ends of the posts and the upper sides of the stulls are not cut at all in this case, provided that the timbers are sawed. Another way to put in square sets is to have them properly framed at the ends and of such a width as to approximately fit the width of the stope, and drive lagging between the square set and the wall to tighten the frame and support the walls. If the timber is to be removed and used again, this is much the better way, but in

cases where there is no intention of removing the timber, the framed method is to be preferred as being both quicker and giving stronger support to the walls, although not making so neat a job. These methods have been described, presupposing that each square set stands by itself, the sets being only connected loosely with one another by the lagging. It is sometimes necessary in wide stopes to frame two or more square sets so that they fit together to span the distance between the walls. In these cases, ties are necessary besides the caps, sills, and posts, the ties being similar timbers to the others, and framed so as to join the square sets together lengthwise of the stope and form a good joint.

Fig. 32

39. Square Sets.—When the rich silver deposits of the Comstock lode in Nevada were discovered, the owners found it impossible to remove the entire mass by means of any known form of timbering. The necessities of the case resulted in the invention of the **square-set system** of timbering, which has been improved on, both in framing and strengthening. When square sets are used, mining is usually carried on by overhead stoping. Fig. 32 is a longitudinal section showing the general appearance of the timbering. As fast as the ore is excavated, new sets are added. Fig. 33 shows square sets on the 600-foot level of

Fig. [illegible]

Fig. [illegible]

the Portland mine, Cripple Creek, Colorado; and Fig. 34 shows square sets in a mine at Mercur, Utah. From these three illustrations, a general idea of square sets may be obtained, but the manner of handling the ore after breaking it is better shown in Fig. 35, which is a cross-section of Broken Hill mine, in Australia, and in Fig. 36, which is a horizontal section of Fig. 35. When the weight of material is not too

FIG. 36

great and the deposit is to be removed quickly, square sets may be employed without any subsequent means of support. In case there is excessive pressure and it is desired to keep the workings open for some time, it may be necessary to either fill the space between the sets with waste material or allow the broken ore to remain temporarily between the timbers. If signs of weakening become noticeable, four or

more of the uprights are lined down the sides with planks and filled from the top with waste rock. In case the ore begins to crush the timbers, it is usual to go into the stope and break down sufficient ore to relieve the weight. It has never been definitely decided from which wall it is best to commence the removal of ore, but some mining engineers commence at the hanging wall and carry the timbers up to it, always keeping the base of the timber sets advanced toward the foot-wall until that is reached.

Fig. 36

FILLING

40. The method of filling openings with waste rock may be carried on either by using light timbering and subsequently assisting the timbering by filling the space with rock; or the mining may be done with little or no timbering, the filling being depended on entirely for holding walls in place.

41. Cross-Cut Slicing With Filling.—At the Cabezas del Pasto copper mine, in Spain, the filling system has been used very successfully. Fig. 37 is a cross-section of the deposit, showing the levels and the main drifts; Fig. 38 is a plan of the third level of the mine; Fig. 39 is a longitudinal section of the mine; and Fig. 40 is an illustration showing the order of breaking the material. After exploring the deposit and determining its form and position, the hoisting shaft No. *8*, the pump shaft No. *3*, and rock-shafts Nos. *1* and *7* were sunk, the latter expressly for lowering rock into the

FIG. 37

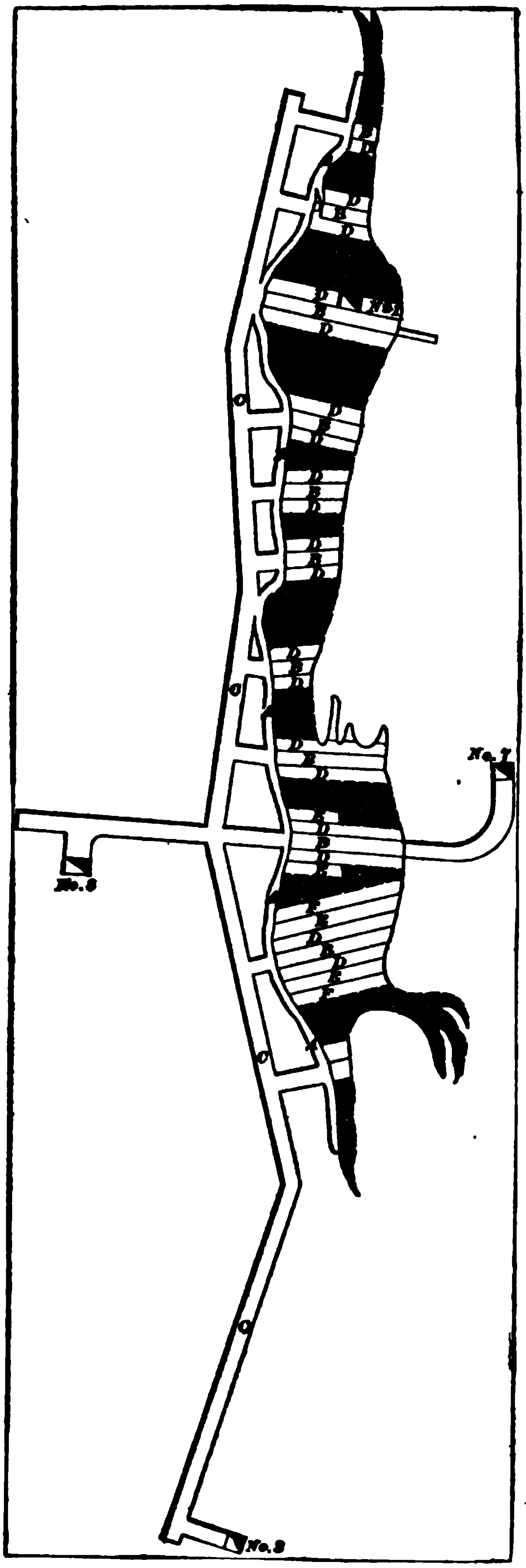


FIG. 38

mine. Cross-cuts were run from the various levels (which were about 65 feet apart) to the deposit, and the drifts *A* driven in the hanging wall of the deposit, following all the changes of direction, and thus determining the exact boundaries of the ore. Cross-cuts were also driven through the deposit from the exploration drifts *A*, and winzes sunk between the levels. The system of working then proceeded as follows: A cross-cut 6 feet high and 6 feet wide was driven from the hanging to the foot-wall of the deposit, as indicated by *B*, Figs. 38 and 40. After this cross-cut was completed, rock was brought from the surface to the next level above and dumped through the winzes into the drift *A* near the end of the cross-cut. The cross-cut was then carefully packed, or filled, with waste rock. The next cross-cuts *D*, Figs. 38 and 40, were driven on each side of the one that had been filled. These were then carefully filled and the cross-cuts *E* were driven and after these

FIG. 8

had been filled, the cross-cuts *F* were driven and filled. This process was continued until a layer of ore 6 feet thick had been taken from under the entire deposit. Next, the drift *A* was also filled and a new one of the same dimensions driven immediately above it. While filling the drift *A*, chutes were provided through which the ore from the new drift could be thrown down to the cross-cuts connecting with the main haulage *C*. These chutes were lined with stonework built of large blocks without mortar. After the new drift above *A* had been completed, slices were taken from the ore by driving and filling the cross-cuts *b*, *d*, *e*, *f*, as shown in Fig. 40. A second cross-cut was not started until the one next to it had been carefully filled. It was found that the

FIG. 40

cost of driving the first cross-cut *B* was one and one-half times as much as that of driving the succeeding cross-cuts of the same level. This was due to the succeeding cross-cuts having two free faces, thus making the blasting more effective. In the succeeding slices, all cross-cuts after the first had virtually three free faces, and hence the cost of mining was less than for driving the first cross-cut *B*. The filling material was quarried from the top of a hill near the mine, but none of the small material from the quarry was put into the mine because it was more expensive to handle underground and it could not be packed so as to prevent the roof settling. The larger pieces could be so packed that little settling would take place. The rock for filling was brought from the quarries in small cars to the rock shaft, which was

Fig. 41

155—30

FIG. 42

provided with balanced cages that were controlled by a drum with a brake only. The loaded car descending on one cage drew up the empty car on the other cage, hence no expense was incurred for hoisting by power. After the quarry cars had reached the level above the one on which rock was required, they were trammed to the top of the rock chutes and dumped, the empty car being returned to the cage and hoisted by the descent of the loaded car. Several levels of the mine were attacked simultaneously; and, as the filling in the upper

FIG. 43

levels became firmly packed, no difficulty was experienced in removing the entire deposit, although both walls of this deposit were weaker than the vein material. The deposit to which this method of mining is applied is about 500 feet long and varies from 20 to 75 feet in width, the average width being about 32 feet.

42. Longitudinal back stoping with filling may be carried on where the ore is hard and tenacious, while the walls of the deposit are weak and friable. The deposit illustrated in Figs. 41, 42, 43, and 44 was first worked as an

open pit, but this system of working had to be abandoned on account of the sides caving; Figs. 41 and 42 are cross-sections of the deposit. Fig. 41 shows one of the main shafts through which the ore is removed. Fig. 42 shows a rock chute, or raise, along the foot-wall through which rock for filling is brought down from the open cut above. In Fig. 41, it will be noticed that there are some raises between the working shaft and the deposit; these are put in as ladder-ways, through which the men could escape in case of fire in the main shaft. Fig. 43 is a longitudinal section showing

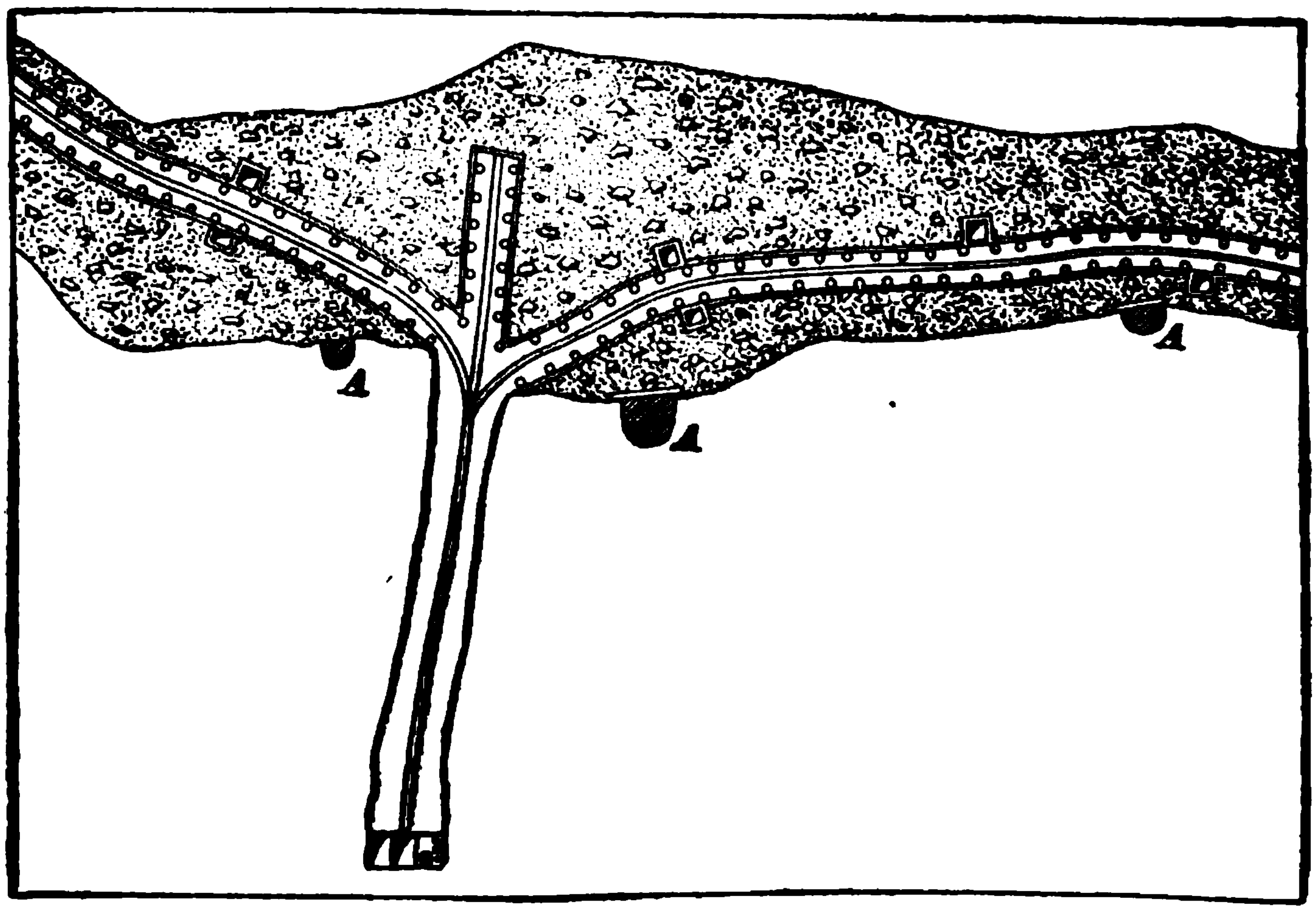


FIG. 44

three levels of the mine. Fig. 44 is a plan of the fifth level of the mine, showing the drifts timbered in the filling, together with the raises, or mill holes, through which the ore is thrown down to the level for tramming to the shaft. The raises *A* are made for dumping rock filling to the levels below. The mining is carried on as follows: A cross-cut is driven from the shaft to the deposit, as shown on the eighth level, Fig. 41. After the cross-cut reaches the ore, a stope 15 or 20 feet in height and the entire width of the deposit is carried along for some distance, and then a drift is timbered, as shown on the seventh level. Cribs are also built at the

sides of the drifts to form rock and ore chutes, after which rock is brought down and the stope is filled nearly to the back. About 10 feet of the roof is blasted down, broken up, thrown through the chutes to the level below, and trammed out to the shaft. Another layer of filling is then put in, after which the stope is again ready for blasting. The work continues in this manner until all the ore has been removed to within a few feet of the next level above. No timbering is used during this work, except in the drifts at the bottom of each level and in building the chutes, which are placed about 30 feet apart, in order to avoid long tram roads.

Fig. 45

When a stope has been worked out nearly to the level above, it is filled and left until work is suspended on the next upper level, then the ore left as a floor between the two levels is recovered from drifts driven in the foot-wall and opposite the floor to be removed, as shown at *B*, Fig. 42. The floor usually caves down on the filling, and as a consequence a little timbering has to be used during its removal, but on account of the broken and shattered condition of the ore and the fact that little or no blasting is required, the product per man may be greater than in the largest stopes.

After a level has been entirely worked out and the floor above robbed, the rock filling may be drawn from it and

used to fill the lower levels. Fig. 45 shows the arrangement at the bottom of the chutes. It will be seen that a special iron apron is so placed that by lowering it any material in the chute will slide into the car, while the flow of material can be stopped by raising the apron. In case the lining of the chute becomes so badly damaged that it requires renewing, it is possible to avoid relining it by converting the ladderway into a chute.

When this system of overhead stoping with filling is used, the stopes are sometimes worked in steps; that is, the first cut is carried forwards some distance, the drifts and chutes timbered, and the filling put in. After this, the first cut may be advanced still farther, the material being trammed back through the timbered drift. At the same time, the second cut is started and the material thrown down through the crib chutes. (After both of these steps have been advanced some distance, the timbering in the first cut, and the filling on both steps, or levels, may be continued.) In this way, the work progresses in benches, as in overhead stoping. The filling material is drawn from the chutes into a car, the box of which is so pivoted as to dump either forwards or to the right or left. In this way, practically the entire filling can be put in with only one setting of the track over which the car passes. That is, the material dumped to the right and left of the track will fill to the walls; and the material dumped in advance of the track provides for the continuation of the work.

43. Room Mining With Filling.—Fig. 46 illustrates a method that was introduced at the Chapin mine in Michigan; it consists in mining a room 20 feet wide across the deposit. This room was carried up to the level above, the excavation being filled with rock dumped through the winzes, as shown in the illustration. Two mills were cribbed up in the filling, one for the ladderway and the other for an ore chute. After the rooms reached the upper level, the pillars between them were drawn by working in the same manner. The ore in this mine was a rather soft hematite, and hence had to be

mined quickly to prevent caving. The expense of quarrying and transporting the material for filling, together with that of building the cribs and keeping the drifts open, has forced the abandonment of this method among the iron ore mines of the Lake Superior region, though there is no reason why it could not be applied to deposits of material having a slightly higher value per ton. At the Chapin mine, this system is applied only to those locations where there had been no settling of the material.

In removing the pillars from the upper portion of the mine and the bodies of ore that have been undercut in the previous work, a system very similar to that described in connection with the Spanish copper mine was followed. Cross-cuts were run through the pillars to the hanging wall; and from these, drifts were run right and left, using light timbering to support the material. When these drifts were completed, they were filled and other drifts run beside them. This process was continued until a slice had been removed from under the entire pillar or body of ore being worked. After this, a new cross-cut was driven on top of the filling and the process repeated.

FIG. 46

44. Advantages and Disadvantages of the Slicing-and-Filling System.—The advantages of the filling system are that the entire deposit is removed. Several levels can be operated at the same time. If operations are being

carried on at a considerable depth, there will be little disturbance at the surface. Little or no timbering is required. The disadvantages of the system are the expenses connected with breaking, transporting, and stowing the filling, which will also require a rock-shaft, otherwise the output of ore will be reduced. Where the material is allowed to fall through the chutes, it will require an extra handling into the cars in the mine; besides, the chutes will have to be kept in repair.

45. Material for Filling.—In the filling operations carried on in the Lake Superior region of the United States, the filling material has been coarse and fine, just as it came to hand, no attempt being made to carefully stow it up under the roof, as is the custom in Spain. In some cases, the mines have been filled with sand or gravel from the surface. When special systems of timbering are used (such as square sets) to support the workings, the rooms are often filled with material after they are completed and the pillars removed by timbering and filling. In many cases, this is not, strictly speaking, a filling system, but is a practical method of disposing of waste material without the necessity of hoisting it to the surface.

46. Obtaining Filling Material in Western Mines. In many of the Western copper and silver mines, the stopes have been timbered and subsequently filled. The material in some instances was obtained by driving a drift into the hanging wall and timbering it securely. When the drift had penetrated the hanging wall for some distance, a chamber was stoped and the material allowed to cave, after which it was drawn out and used in filling the old mine workings. When the hanging wall is composed of a somewhat friable rock, this method has proved very successful and has furnished filling material at a small cost per ton, at a point near the place where it was to be used. It is not a purely filling system as the ore is extracted by means of timbering, and the filling is used to support the walls while the pillars are being removed. The ore from the pillars is also extracted by the aid of timber sets.

THE CAVING SYSTEM OF MINING

47. Mining Ore Bodies by Caving.—In the long-wall system of mining described, the roof is allowed to cave, the entire deposit being removed. When a bed of ore is thin and at some distance from the surface, the caving will affect the surface very little, but when it comes to dealing with masses of ore the dimensions of which may run into hundreds of feet, it is evident that the surface will be badly disturbed by allowing the overlying strata to cave and fill

FIG. 47

the opening. In the United States, the first attempts to work wide masses consisted in filling the rooms mined out and then drawing the pillars, or a portion of the deposit was worked out and the excavation timbered. Then, after this timbering had been crushed and the roof had settled on the formation below, more of the ore was removed by timbering as before. The caving systems now practiced are derived from experience.

48. Caving Roof or Gob Only (North-of-England System).—The caving system introduced into the United

States from the north of England consisted in the removal of the material from immediately below the top of the deposit. The top was then allowed to cave into the workings and another slice below removed. By this method, the miner always worked under a roof of broken material supported by light timbering. Conditions have developed a system that is exactly the reverse of this; in it, the miner goes some distance from the top of the ore and undermines it, after which the material falls of its own accord into the workings and is removed with little or no blasting. Between these extremes there are many intermediate systems.

Fig. 47 shows a gang of men driving a drift through soft ore at the top of the formation, previous to starting a new slice in the caving system. It will be seen that just sufficient lagging is put in place to keep the drift open. After the drift has been completed, the floor is covered with timbers placed at right angles to the length of drift, as shown in Fig. 48, and the timber sets are removed or blasted down so as to allow the roof to cave in and fill the space. The timbers on the floor form a support for the material from above, and in subsequent operations at a lower level, the rock and waste material are kept out of the workings by simply

Fig. 48

FIG. 49

FIG. 50

supporting this old timber floor. Figs. 49 and 50 show drifts after the roof has caved. Fig. 51 shows the general arrangement of drifts and levels in this system of mining. The main levels connected with the shaft are usually about 100 feet apart and on these levels haulage drifts are driven through the ore, as shown at *d*.

(*b*) (*c*)

FIG. 51

When the ore above the level has been removed, the drift on this level is abandoned and the block of ground between it and the next lower level is worked. The method of working is as follows: Drifts *a* are driven to the walls of the deposit from or near the top of the raise *c*, and from them other drifts *b* are driven and timbered, as shown in

Fig. 47. The size of these drifts depends largely on the character of the ore, and may be taken as about 8 ft. × 8 ft. in the clear. The floors in the finished drifts have three poles laid lengthwise of the drift on which round saplings or sawmill slabs are laid at right angles and then the drifts *b*, Fig. 51, are caved. This operation is repeated until the entire slice of ore is removed.

In Fig. 51 (*a*), which is a plan through the top of the raise *c* and the drifts *a* and *b*, two drifts have been caved on one side of the deposit and three on the other, and two others—one on each side—are ready for caving. Sometimes, the raises are placed close together and the drifts *b* are made short. By this means, a large proportion of the ore can be shoveled into the raise without the use of a wheelbarrow.

49. Caving a Back of Ore.—Fig. 51 (*c*) illustrates the method of caving a back of ore. *d* is the main working level, which has been driven through the ore; *c* is the raise; *a* is the upper drift, from which caving is being carried on; *e* is the next lower or subdrift, which has been driven preparatory to caving the block between it and the level *a*, after the upper slice has been removed. This method of operating does not produce quite as clean ore as the one described, on account of the fact that more or less gravel or waste rock becomes mixed with the ore during caving.

In some cases, where the ore is of a somewhat harder nature, a slight modification of this system is used; and in place of driving the drifts *a* at the top of the portion of the deposit to be worked, they are driven several feet below. The drifts *b* are on the same level, and hence under some ore, which is allowed to cave to the miner as he works back toward the raise. After one of the side drifts has been caved, the work progresses, taking off successive subdrifts until the entire slice of the deposit is removed.

50. Example of Caving a Back of Ore.—Figs. 52, 53, and 54 illustrate the method of laying out a mine for the caving system just described. Fig. 52 is a vertical section on the line *AB* in Fig. 54. Fig. 53 is a vertical section

through shafts *4* and *5*, also on the line *A B*, Fig. 54. Fig. 54 is an outline of the ore body as it appeared on the eighth level. All the shafts were originally vertical, but as the caving progressed, the upper portions of shafts *2* and *3* were carried away with the caving ground, and inclines were put down that intersected the old shafts at points below the caving. This can be seen in the case of shaft *2*, Fig. 52, where the incline intersects the old shaft between the fifth and sixth levels. In the upper portion of the mine, the main levels were driven 75 feet apart, and generally there were two main drifts at the bottom of each block of ore. From

SOUTH

FIG. 52

these main drifts, raises were put up at intervals of about 50 feet, and from these raises, four series of subdrifts were run. The timber sets in the main drifts have caps 9 feet long and legs 7 feet long, while those in the subdrifts have 6-foot caps and 6-foot legs; this leaves about 8 feet of ore between the sublevels. The timber sets are placed from 3 to 4 feet apart in the drifts. As the raises are put up, timber sets are placed at the first subdrifts, but these are not worked at once, being left to strengthen the main drifts until the fourth, third, and second subdrifts have been worked out. When the subdrifts have been completed, the block of ore

lying between any two levels is honeycombed with drifts with vertical intervals of 8 feet of ore. When mining above has been completed, the removal of the ore pillars on top of the subdrifts begins. The pillars are sliced away and the back

FIG. 58

is caved as illustrated in Fig. 51 (*c*). The ore is removed in wheelbarrows to the chutes leading to the main level below. When waste rock or overlying timbers appear, a new slice is taken off and the ore caved as before, until finally all the ore above the level of the subdrift has been worked out,

after which the operation of caving is continued on the next block below, which, in the meantime, will have been honeycombed by the subdrifts.

There are two objections to this method of work: (1) The subdrifts are so close together that sometimes the crushing of the first subdrift above the main drift will crush the main

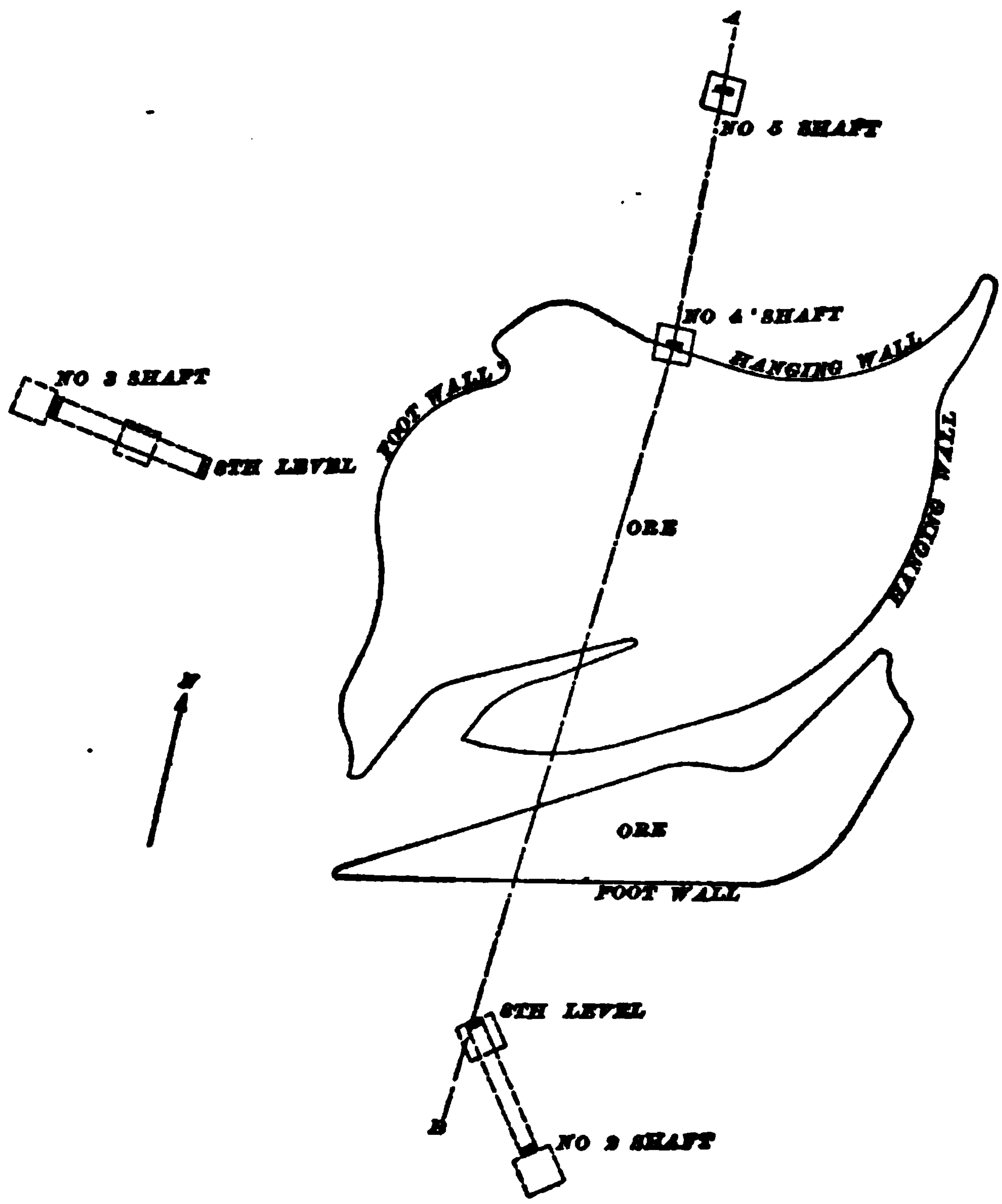


FIG. 54

drift. (2) The ore obtained on any sublevel has to be handled twice; that is, with wheelbarrows to the chutes and subsequently in the cars to the shaft.

51. Modification of the Caving System.—A method that overcomes the difficulties in the system just described is illustrated in that portion of the mine below the eighth level, Fig. 53, which is laid out on a slightly different plan.

What are called *intermediate main levels* are driven through the ore at intervals of 20 feet instead of 75 feet, and no sub-drifts are used. The intermediate main levels are of the regular size, the timber sets having 9-foot caps and 7-foot legs, which leaves about 10 feet of ore to be caved instead of 7 or 8 feet, as in the previous case. Stations are made at the shaft for each intermediate main level. Under this modified system, the removal of each 20-foot block of ore may be accomplished as before, but the labor of driving raises will be saved and cars can be used in the drifts, thus doing away with most of the wheelbarrow work. In some cases, the rock and sand from the caving ground will become more or less mixed with the ore, and thus reduce its value; but as the ore is obtained very cheaply, it is thought that this may compensate for the injury to its quality.

52. Caving All the Ore Between Two Levels.—The plan of leaving a back of ore above the working level to gradually cave and fall to the level with little or no blasting, has been expanded until the system illustrated in Fig. 55 has been developed. The deposit is divided, by levels and cross-cuts, into blocks from 200 to 250 feet in length, which are worked as follows: Levels are driven 100 feet apart in the hanging wall *H*, parallel with the deposit and usually about 20 feet from it. After all the material in the upper level has been worked out and the surface caved to this point, the cross-cuts *B* are run from level *A* through the ore deposit to the foot-wall; and from them, two raises *D* are driven nearly to the level above. The tops of the raises are connected by the drift *E*, after which the slice of ore *a*, *b*, *c*, *d* is removed by underhand stoping for 8 feet along the deposit (as shown at *a e*); thus the ends of the large block are cut free from the adjacent material, with the exception of the portion left above the cross-cut *E*. While this work is going on, the cross-cuts *C* have been driven through the ore and the entire mass, with the exception of the pillars *N*, has been undercut to a height of 7 feet. After this work is completed, the pillars are reduced as

much as is consistent with the safety of the miners who are doing the work and are then blasted out.

Fig. 55 illustrates the method of preparing a block of ore for caving. A portion of the hanging wall has been cut away in such a manner as to expose the timbering of the drift *A* and of the cross-cuts *B* and *C*, also the completed cut at the left-hand end of the block of ore; the small pillars supporting the ore can be seen at *N*. At the right-hand portion of the illustration, the ore has been cut away in such

FIG. 55

a manner as to expose the cross-cut *B*, the raises *D*, and the cross-cut *E*. The foot-wall can also be seen at *F*, and the downward continuation of the ore body at *P*; *G* is the caved ground above the ore. The entire block of ore settles so as to fill the space undercut. After several months have elapsed, the material is usually crushed so fine that the greater part will pass through a 3-inch hole. When this has taken place, the cross-cuts *C* are reopened through the crushed ore and are strongly timbered. From these cross-cuts *C*, timbered drifts are driven to the ends of the block; and from the drift nearest the foot-wall, short cross-cuts are driven to the foot-wall. The drifts in the caved

ground are placed about 25 feet apart, and the ends or faces of the drifts are constantly full of broken ore, which slides down and into them (see Fig. 47). The work of mining is carried on by shoveling the ore into mine cars and taking it to the shaft. When a full output is desired, the force at any face consists of one miner and four muckers; two of the men are always employed in loading ore, while the other two are tramming it to the shaft. No drilling is required, and but little powder is used in blasting. Drawing the broken material in this way forms funnel-shaped spaces in the broken ore, which are eventually filled by the timber and waste material above. When this gob appears at any working face, the miner pulls or blasts out a few timbers and draws ore from a point nearer the main cross-cut. This operation is continued and repeated until the ore in the territory formerly transversed by these drifts has been exhausted and replaced by gob, after which ore drifts from the main cross-cuts are driven in the ore that was left between the first drifts. In actual work, it is found possible to draw the ore for 3 or 4 feet on the sides of these drifts and, consequently, a second set of drifts practically cleans up the broken ground. After the ore has been removed, the gob settles firmly into the place formerly occupied by the ore body, then the adjoining block can be undercut and separated from the ore at its solid end, after which it will be allowed to cave and will be removed in the manner just described. One advantage of this method of caving is that the output of a mine can be greatly increased if there are a number of blocks of caved ore ready for work.

53. The Safety of Men in Caving and Filling. Caving-and-filling mining is comparatively safe, for the miner is always close to the back of the deposit, and hence knows the condition of the material over him. The miner knows also that the ground above is working, and hence only attempts to keep those passages open that are actually required in mining. In some mines, after the caving has commenced, the air pipes are removed and the use of air

(a)

FIG. 58

(b)

drills prohibited, since where they are used to cave the back of a formation, the miners are liable to drill too deep holes and to use such large quantities of explosives that they bring down more ore than they can handle, and hence lose control of the ground.

54. Rooming and Caving.—Owing to the fact that where a back of ore is caved more or less gob becomes mixed with the ore, and also to the fact that where only the waste is caved but a small output is obtained, some engineers have adopted a compromise between the old system of room mining with square sets, and the caving system, and from this has been developed a system that has been employed to some extent, in which the rooms are timbered with what are called *saddlebacks.*

An attempt has been made in some mines to lessen the amount of timbering by using what are called *stull rooms.* Fig. 56 (*a*) shows a cross-section or end view and Fig. 56 (*b*) a vertical longitudinal section through one of these stull rooms after the upper and lower drifts have been driven, but before the ore has been mined out. The level *A* is driven something over 60 feet below the top of the formation. The raises *B* are then put up and the large drift *C* is driven and timbered with saddleback sets as shown. After this, the room is worked to the dimensions indicated by the vertical dotted lines, the ore being milled through the raises and trammed through the level below the shaft. When the room has been worked out, the pillars may be attacked at the end farthest from the main drift, beginning usually near the top, the ore being replaced by light timbering. Work is continued until the roof shows signs of crushing, when the miner leaves the working place while the roof crushes in and fills the rooms, together with the space left where a portion of the pillar has been removed. When the caved material has thoroughly filled the old workings, the pillars may be attacked by driving drifts through them to the end farthest from the main working drift, and then taking off subdrifts from this inner end of the pillar, supporting the roof with

light timbering until it shows signs of caving, when the men again leave the work until the timber is crushed.

55. Height of Rooms Where Roof Is Caved and Method of Recovering Pillars.—While the method of mining just described produces ore very cheaply in the rooms, there is danger of losing a portion of the high pillar. This remark applies equally well to high (seven to nine set) rooms timbered with square sets; and on this account, the distance between working levels in which it is intended to allow the roof to cave are being reduced. Rooms are frequently driven but three sets high; and after they are completed (from 60 to 100 feet long), the pillars between the rooms are attacked at the end farthest from the main haulage drift, light timbering being used to support the roof. After several sets or subdrifts have been removed from the inner end of the pillar, it will usually show signs of crushing; then the miners blast the timbering that has been placed to support the roof while robbing the pillar, together with a few sets at the inner end of the stope or room. The roof then caves in; and after it has packed, a drift is driven through the pillar to take off several more subdrifts or sets. When the roof again shows signs of caving, the miners leave the workings until they have filled. This method of procedure is not so slow and dangerous as it might appear from the description; and when the work is done rapidly, considerable advance can be made before a cave occurs. When the light timber used begins to crush, it is sufficient warning that a cave is coming.

56. Advantages and Disadvantages of Rooming and Caving.—By the rooming-and-caving method, the pillars are crushed and less blasting is required while they are being robbed. Also, practically all the ore in the mine is recovered with the use of little timbering. When any portion of the deposit is attacked, the work is pushed as fast as possible until there are signs of caving. The advantages are that the ore is obtained cheaply and comparatively little timbering is used.

The disadvantages of this method are that only the top of a deposit can be attacked, and the surface is allowed to cave into the mine, thus requiring the machinery and buildings to be placed some distance from the workings.

57. Quality of Ore Obtained by Rooming and Caving.—By working low rooms and then robbing the pillars with the use of light timbering, fairly clean ore is shipped, and in this way a better price may be obtained for the product of the mine. The increased price may more than pay for the difference in the cost of working, by means of light timbering, with subsequent caving, and that of caving a back of ore.

58. General Remarks on Mining Methods.—No general rule can be laid down as to the best method for working large deposits, but the following remarks cover some points that are to be considered:

If the surface is of greater value than the ore, the caving method is out of the question.

The character of the ore will, to a large extent, determine the method of mining; for if it is extremely hard and breaks like quartz or hard-iron ore, it may be impossible to employ any of the caving systems. In such a case, it will be necessary to adopt one of the filling methods, or to support the openings by means of square sets or some other form of timbering while the material is being removed.

When the deposit contains portions of ore that are of a considerably lower grade than the main portion, pillars of this lean ore may be left to support the hanging wall while the remainder is being removed. Some gold, silver, and copper mines are worked in this manner, without the use of timber, by simply leaving pillars of lean ore to support the roof.

The character of the walls, and especially that of the hanging wall, will have an important influence on the method of mining to be adopted; for if the hanging wall is composed of material that tends to form a natural arch (such as strong limestone), it is unsafe to attempt to cave the material, as the roof will arch for some time and finally cave and crush everything before it.

IRREGULAR DEPOSITS OF A POCKETY NATURE

59. Drift-Set Mining.—Deposits of a pockety nature are usually mined by means of drift sets placed as soon as the excavation is advanced sufficiently. Fig. 57 is a section through a zinc deposit, where the ore occurs on limestone under clay, the surface of the limestone being covered with projections (which are called *chimneys*) and the ore occurring not only in the hollows between the chimneys, but, as a rule, in uninterrupted layers of varying thickness over the entire surface of the limestone. To obtain the ore, small shafts are sunk to the lowest part of the deposit and from them drifts and inclined raises are made to follow the ore.

FIG. 57

When the layer of ore is not as thick as the height of an ordinary drift set, it is mined by using posts and caps.

Fig. 58 (*a*) is a section showing the drift sets used at the sides of the deposits and the caps and posts used in working over the top. Fig. 58 (*b*) is a section on the line *A B*, Fig. 58 (*a*), and shows the manner in which caps are supported on the props. The caps are simply slabs or pieces of plank that, with the props, serve to hold up the clay while the ore is being removed. The work of removing the ore begins at the top of the deposit and progresses toward the bottom of the shaft. On the rise of the chimneys and in the lower part of the deposit, drift sets are used; and when the ore is too wide or too thick to be removed by one series of drift sets, two or more are placed side by side or

on top of each other, as shown in Fig. 59. In this way, the entire body of ore is removed, and any clay that is encountered in the drifts is used as stowing. Each shaft serves a comparatively small area of the deposit, and in this way the

(b)

(a)

FIG. 58

length of the underground haulage is reduced. The drifts are crooked, as they follow the irregularities of the limestone. The ore is brought to the shaft in specially con-

FIG. 59

structed wheelbarrows, which have deep boxes, and is dumped into iron buckets and hoisted out of the mine. The workings from one shaft are continued until they encounter those from the adjoining shaft. The shaft linings

employed are 2- or 3-inch planks; and as the clay will usually stand in place for a short time without any support, it is possible to recover most, if not all, of the lining material, when a shaft is abandoned. The greater part of the timber used in the drift sets is lost.

SPECIAL METHODS

60. Mining Low-Grade Deposits.—Salt and similar materials having a low value per ton must be mined by the least expensive method possible; hence, little or no timbering can be used. In all salt-mining operations, it is of great importance that the formation shall not settle or cave in such a manner as to crack the overlying material, and allow fresh water to flow in on the salt. The mistake of working too close to the surface was made in the first salt mines in Louisiana and practically caused a loss of the greater part of the deposit above the mine, for fresh water flowing in through the roof dissolved the salt and finally stopped operations.

61. Room-and-Pillar System Applied to Mining Salt.—Salt is usually mined by some modification of what is called **square work**, or the **room-and-pillar system.** When a deposit of rock salt is overlaid with wet formations, the salt must be carefully secured against the influx of surface waters. After sinking a sufficient distance into the deposit, drifts are run from the shaft, and from these, chambers are started. The salt is usually worked out by undercutting with a pick so that it may have two free faces and be barred or shot down.

Fig. 60 is a cross-section of a portion of a deposit being worked by the chamber system. Fig. 61 is a plan showing the arrangement of the pillars that are left to support the roof. First, an undercut *A*, Fig. 60, 7 or 8 feet high and the full width of the intended chamber is driven for 200 or 300 feet. The salt is trammed out as fast as it is broken down. After this room is completed, the roof is blasted down until a chamber 18 or 20 feet high is excavated, as

illustrated at *B*, Fig. 60. The salt is allowed to remain in the chamber for the men to stand on while enlarging the height of the room. A part of the broken material is then

Fig. 60

removed, after which the men stand on platforms to drill holes in the roof and blast down another layer. Then, by standing on shorter platforms placed on the salt already

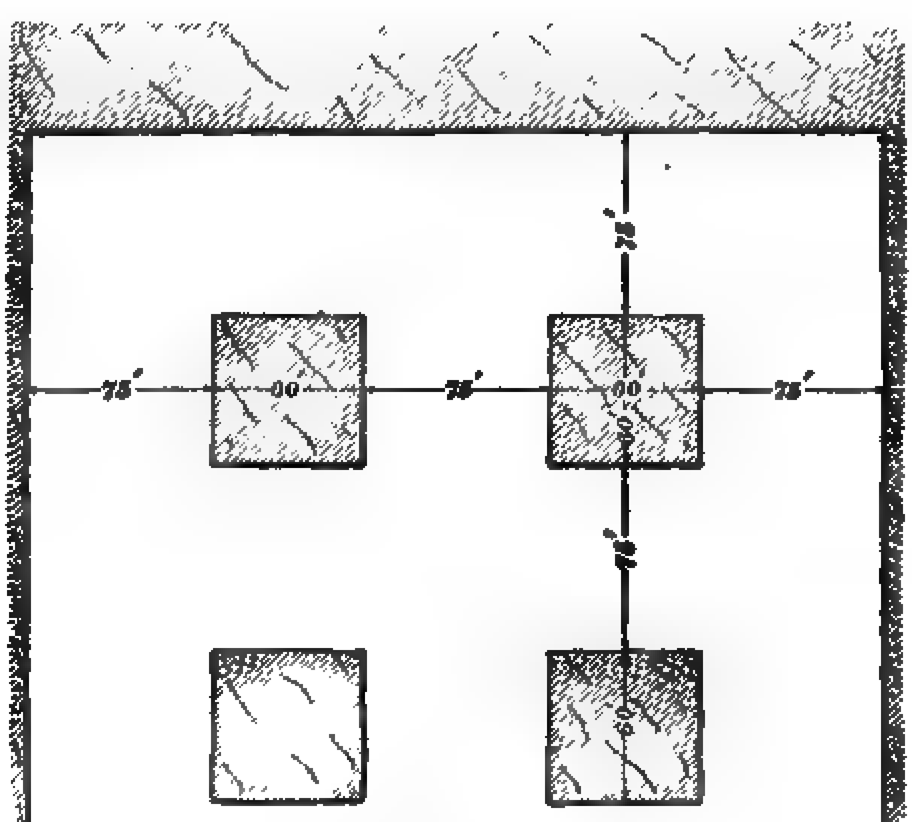


Fig. 61

thrown down, they again attack the roof. Owing to the fact that the broken material takes up more space than when in the solid, the chamber is soon so nearly filled that the workmen can stand on top of the broken salt and break down

the roof. This is continued until the center of the room has been carried to the desired height. The chambers are arched at the top to give them a strong roof; and before the salt is removed from the room, the miners carefully examine this to see that there are no loose pieces that might fall and endanger the lives of the men working below. After the chambers have been driven in one direction, cross-cuts are broken through between them in such a manner as to leave pillars nearly as wide as the chambers themselves. The roofs of the chambers form the floors of the chambers above.

62. Portion of Deposit Recovered.—By the square-work method, scarcely one-half of the deposit can be removed; but the advantages are that no timbering is required and the material is broken down at a very small expense. After all the salt that can safely be removed by this means has been taken from the deposit, the mine must be abandoned; but it can be allowed to fill with water, which soon becomes a strong brine; this may be pumped out and evaporated, if it is so desired.

An advantage in mining salt in this way is that coarse rock salt, for which there is a demand, can be obtained. One chamber 200 feet long and 75 feet wide, with a mean height of 65 feet, represents about 50,000 tons of rock salt; hence, it will be seen that if the deposit is large, even this wasteful method of mining may produce great quantities of the article. When rock salt occurs in comparatively thin seams between firm rock walls, a greater percentage of the deposit may be mined with safety. At times, it may be necessary to use some timbering.

63. Room-and-Pillar System Carried to Great Depths.—When the room-and-pillar system is carried to a great depth, it becomes necessary that the lower pillars be of greater cross-section than the upper ones. Therefore, with each succeeding level, less and less of the deposit is recovered, until utimately a depth is reached where it would be unprofitable to mine the small chambers that could be left unsupported with safety.

Fig. 62 shows a cross-section of a deposit worked by the room-and-pillar system. It will be seen that the chambers become smaller and smaller with depth owing to the necessity of leaving larger pillars on each succeeding level. The dimensions given in Figs. 60, 61, and 62 are not general, and would have to be varied to suit the deposit being mined.

64. Salt Mining With Machines.—Salt is sometimes mined by using undercutting machines, such as are shown in Figs. 63 and 64. Fig. 63 is termed a *pick machine* and is

FIG. 62

worked by compressed air; Fig. 64 is a *chain-cutter machine* that is worked by electricity. Both classes of machines are effective, and have a wide use in coal mining. There are special undercutters made for long-wall mining, and special cutters for shearing, that is, making vertical cuts. Where the seams of salt are between rock formations and are comparatively thin, the undercutter may remove the rock from below the salt; while, where the deposit is very thick, the machine is used simply for making an undercut in the salt. Machines may also be used in driving the first undercut shown in Fig. 60, at the bottom of the deposit, the balance of

the 7 or 8 feet being mined by drilling holes near the top and blasting down the salt.

65. Robbing the Pillars.—When the room-and-pillar system of mining is applied to large bodies of ore and it has only been possible to remove comparatively small portions of the deposit, attempts have been made to remove the floors and pillars left in the mine. In some cases, the rooms have been filled and a portion of the pillars worked away,

Fig. 68

but this made it necessary to resort to one of the other systems of mining—as, for instance, the filling system or the caving system. In the case of some large copper deposits in Spain that were worked by this method, it was found that less than one-half of the ore could be recovered, therefore, the overburden was removed and the pillars and floors worked from open cut. In this case, it would have been much better to have removed the overburden in the first place, but there may be cases in which the mine has to pay a great part of the expenses as the work progresses, and as the removal of the overburden would be very expensive at first, the method referred to may have been necessary.

Fig. 64

66. The Room-and-Pillar System in Comparatively Thin Seams.—When room-and-pillar mining is applied to seams, not over 20 feet thick, that are situated between hard rock walls, it may be possible to leave only pillars of ore in the mines. If the pillars and galleries are of the same width, this method will remove practically eight-ninths of a vein, but, when it becomes necessary to leave a floor or a roof of the mineral, the amount won is less.

67. Leaching Salt.—Deposits of salt that are so located that it is impossible to mine them with profit may be worked by drilling a large hole into the formation and lining it with a casing pipe. A smaller pipe is then let down inside as close to the bottom of the hole as possible, while the casing is usually stopped at or near the upper portion of the deposit. Water is allowed to flow down through the annular space between the outer and inner pipes. This water dissolves the salt from the formation and a strong brine settles at the bottom, whence it may be pumped through the inner pipe. By this method, a large amount of salt has been removed from some deposits, the brine being subsequently evaporated.

Another system of leaching is to bore several holes into a salt deposit and let water flow into them. After a while, a deep-well pump is used to draw out the brine formed. If water is allowed to flow into the holes not being pumped, an underground connection will soon be made between them and the pump, making the operation of pumping and saturation continuous.

68. Mining Sulphur by Heat.—In some locations, there are sulphur deposits at some distance from the surface and so located that regular mining methods would be practically impossible, on account of the soft and treacherous nature of the ground. Attempts have been made to work such deposits by putting down pipes similar to those used for leaching out salt deposits as just described, but in place of sending down water, superheated steam was used to melt the sulphur, which was then pumped out through the inner

pipe. On account of sulphur melting easily at 114° and forming a thin straw-colored liquid this plan was thought feasible; but at 200° sulphur becomes viscid and will not flow at all, a fact overlooked in Louisiana where this process was tried. Again, the pump valves clogged with sulphur, which cools rapidly, so that the plan was a failure.

VENTILATION OF ORE MINES

69. Importance of Ventilation.—Because the ventilation of ore mines is not imperative, as is the case with coal mines, too little attention has been given this important feature in mining. While it is extremely rare that air is forced through ore mines to drive out explosive gases, the fact remains that there are other gases almost as detrimental to the health of miners as explosive gases. Usually, the importance of ventilation in metal mines increases with the number of men employed and the size of the workings. In driving long tunnels, raises, or winzes, or even in overhead stoping, the air will become very bad unless some artificial method of ventilation is employed.

70. Pollution of air in mines may be from the following causes: Respiration of the men; exhalations and excrement from their bodies; gases arising from the combustion of oil or candles used for lighting purposes; absorption of the oxygen from the air by minerals oxidizing; and the gases that may arise from the oxidation of minerals; decay of timbers; gases resulting from the explosion of gunpowder, dynamite, etc.; and last but by no means least in some mines, the dust arising from drilling and handling minerals.

71. Respiration of Men.—In breathing, men inhale oxygen from the atmosphere and exhale carbon dioxide; their bodies also give off exhalations of other organic gases. Men doing hard manual labor, as miners, require more air than is allowed per person when figuring on the ventilation of halls, schools, theaters, etc.; hence, the rules used in such

cases become useless when applied to problems in mine ventilation. The average man, when engaged at hard work in a mine, requires about 30 cubic feet of pure air per minute; and the horse, 150 cubic feet per minute for respiration. Assuming that a man breathes twenty times per minute, each breath being 40 cubic inches, he will exhale 28 cubic feet of gas per hour, which will vitiate 1,900 cubic feet of air per hour. He will emit, besides the air from his lungs, a quantity of vapor that has been estimated as .0836 pound of water per hour, enough to saturate 7.1 pounds of air at 60°. To be healthful and pleasant, the air should be only one-half saturated, hence 14.2 pounds of air, or 187 cubic feet added to 1,900 cubic feet, gives a total of 2,087 cubic feet per hour, approximately as stated 30 cubic feet per minute. Owing to the fact that actively poisonous gases are not often met with in metal mines, and that animals are rarely put under ground, the metal miner often gets along with much less than this amount.

72. Gases Resulting From the Lights.—When lights (either candles or lamps) do not smoke, the gas produced is practically all carbon dioxide. The lights may produce from two to four times as much of this gas as that produced by the breathing of the men, hence, more fresh air must be provided for the use of the lights than for the men. Oil lights often smoke; and this smoke is more objectionable than the gases produced by the lights.

Candles are used in most metal mines, though lamps may be required where large stopes are being worked. Taking 30 cubic feet per minute for respiration and 60 cubic feet per minute for candles, the total quantity of air that is required for sanitary purposes is about 100 cubic feet per minute per man, a quantity about one-tenth of what some writers claim is needed in coal mines giving off firedamp.

73. Gases From Minerals.—It is not uncommon to find in mines that gases are given off from minerals and from rocks that enclose the excavation. When ventilation is sluggish, the absorption of oxygen by pyrite, or by iron

minerals passing to a higher state of oxidation, is sometimes very marked. Sulphur dioxide and hydrogen sulphide may be formed under certain conditions. Carbon monoxide and carbon dioxide may be given off from rocks in small quantities, which may become injurious when ventilation is sluggish, particularly in lead, zinc, and copper deposits worked in limestone. Poisonous arsenic, phosphorous, and mercurial gaseous compounds, as well as marsh gas, have been detected in ore mines.

74. **Effect of Explosives.**—The foul air that is produced by the burning of the explosives is another factor that enters into the problem of ventilating ore mines. Ordinarily, the miners have to wait for the smoke to drift from the point where the blast was fired or to slowly diffuse itself through the air in the mines before returning to their working places.

Black powder produces less objectionable gases than those resulting from the use of giant powder (dynamite) or any of the nitroglycerine explosives. All explosives give off a certain amount of foul gas, and particularly when imperfectly burned. It occasionally becomes necessary to experiment before the mine authorities can determine which explosive will give the best results for their mines. Ordinary gases given off from the explosives are carbon dioxide, nitrogen, carbon monoxide, and hydrogen; some of the high explosives give other compounds. Nitrogen is the inert gas in the atmosphere, and neither it nor the carbon dioxide is poisonous, though neither will support life. Carbon monoxide is a highly poisonous gas and the hydrogen is, if anything, slightly poisonous. Explosives that produce nothing but carbon dioxide and nitrogen are the least objectionable, and those that produce the maximum amount of carbon monoxide are the most objectionable. Some explosives also produce various sulphur gases or fumes, and dynamites send into the air about 25 per cent. of their weight of solid matter.

75. **Decay of Timbers.**—Another cause for the pollution of mine air to be considered is the decay of timbers.

This is not only a deoxidizing effect, with the evolution of carbon dioxide, but the timber may putrefy, resulting in the emanation of various noxious gases. When timbers left in an old mine decay, they have been known to produce explosive gases, which, when the workings have been broken into, have rushed in on the miners, sometimes producing disastrous explosions. In many instances, the removal of decaying timber cannot be accomplished; but so long as there is an offensive smell arising from such sources, an additional supply of air is needed to prevent injury to the health of the men.

76. Dust.—Metal mines are more likely to be very wet than too dry, but there are some that are very dry. The dust resulting from the handling of ore, from drilling, and from blasting in a dry mine is liable to be very sharp and injurious to the lungs of the workmen, if not actually poisonous chemically. For its removal, it must therefore either be damped by water or quickly removed by currents of air. The dust of mines may be actively poisonous, as for example that found in lead-carbonate, arsenical and antimonial gold and silver, and cinnabar mines. Such dust may be removed by good ventilation. The upper levels of a mine may originally have been wet, but the working of the lower levels may have drained them so thoroughly as to render them quite dusty and unhealthy. Miners' asthma results from fine stone dust and lamp smoke, which accumulate in the lungs. It is a fact that coal miners live longer than ore miners, and this is only to be accounted for by the former having better air to breathe.

77. Explosive Gases.—Marsh gas (the principal constituent of the firedamp of collieries) is the most important explosive gas met with in the mines. It is derived from the decomposition of carbonaceous matter not necessarily coal, but might be derived from natural sources such as petroleum, asphaltum products, old timbers, or other similar matter. It may accumulate in the interstices of the rocks, and subsequently in the openings of the mine, in sufficient quantities to produce an explosion when ignited.

Explosive gases are not usually looked for or provided against in ore mines, and yet there are a number of disastrous explosions on record. Marsh gas has been found in salt mines, iron-ore mines, sulphur mines, and in some of the quicksilver mines, particularly in California. It may be stated that when an inflammable or other injurious gas has been discovered in a mine, that an additional supply of air to dilute it is imperative.

78. Underground Temperature.—Below the surface of the earth, the temperature gradually rises. This rate of increase has been variously estimated, and was formerly stated at 1° F. for every 60 feet of descent; but later observations have shown that the increase is generally much less rapid, and 1° in 100 feet would be considered a sharp rise. Of course, there are exceptions to the rule; and some workings in regions that have been subjected to volcanic action or contain hot springs may be extremely hot at comparatively shallow depths. Men have worked where there was a rock-and-water temperature of 150° F., but they had to be supplied with an abundance of fresh air and could only work 15-minute shifts before going to a cooling room. In extreme cases, the intake pipes have passed through ice water (as, for instance, at the consolidated Virginia mine in Nevada).

The temperature of shallow mines, if not raised artificially, is cooler in summer than that of the surface and warmer than the surface in cold weather; usually, it is only at considerable depths that the temperature is higher than the surface all the year around. Where there are marked changes of surface temperature between day and night, it often happens that at night the air in the mines is warmer than the surface; while during the daytime the reverse is the case. All these facts have an important bearing on the natural or artificial ventilation of ore mines, since, unless regulated, the air-current will travel in one direction, and then the reverse, or there may be no current of air traveling at all.

NATURAL VENTILATION

79. In the great majority of ore mines, **natural ventilation** is depended on entirely; and in small mines, it is often allowed to take care of itself, although the adoption of simple cheap arrangements for controlling the natural air-currents would greatly improve the conditions. As mine workings are extended, the connections that are made for developing the ground or for convenience of handling ore waste and water can be used to assist in ventilation. Where mines have a hoisting plant, the movement of the cars or cages will stir up the air and possibly assist natural ventilation.

In case there is an air-compressor plant for air drills, the exhaust from the drills will afford some slight aid to natural ventilation, particularly at the face where drills are worked. Even when starting a mine in ordinary rock, without knowing in advance to what extent the workings are to be carried, a small compressor plant is sometimes set up so that the use of air drills may expedite the work of development. Too much importance is given to the ventilation of workings by air drills, as they are only serviceable where used and not very effective there. It is usually best to avoid putting in fans or blowers, together with the connecting pipes, unless this becomes absolutely imperative; this is especially true in a case where the mine has no other machinery. However, good air must be furnished the men if good work is wanted. Using gas or gasoline engines below ground to drive blowers is bad practice, even when the workings are not deep, as carbon dioxide from the exhaust will invariably go to the lowest point in the mine.

80. General Principles.—The theory of mine ventilation is based on the following principles:

1. Air heated above the temperature of the surrounding atmosphere at a given level becomes expanded and therefore lighter and has a tendency to rise; cooler air sinks for the opposite reason, and thus a current of air is established.

2. **Diffusion** is the tendency of two or more gases of different densities, but originally of like temperatures, to become uniformly mixed, without regard to difference in weight.

3. **Convection** is the tendency of currents of different temperatures to seek an equilibrium of temperature, and in the circulation thus produced to approach the uniformity of temperature within a closed space.

The first principle explains the movement of the main currents in the mine, for the air becoming warmer has a tendency to rise and thus allow a supply of colder air to take its place. The greater the differences in temperature, the more rapid will be the movement of these main ventilating currents. Diffusion and convection together explain why powder smoke and foul air, in the absence of appreciable ventilating currents, slowly become diluted through the mine air, so that while the whole body of air is vitiated, that at the place where the blast was fired or the foul air produced becomes in time diluted enough to be breathed.

81. Weight of Air.—Natural ventilation depends on the difference in temperature between the air outside and inside the mine; in other words, the difference in weights between two columns of air having different temperatures. It has been ascertained by experiment that the weight of 1 cubic foot of air at 32° F. and under a barometric pressure of 30 inches, is .08073 pound. Air is not so heavy at high altitudes as at sea level, and its weight changes considerably.

82. Atmospheric Pressure.—The **atmospheric pressure** is the pressure due to the weight of the atmosphere. It is measured by the barometer, and this gives rise to the synonymous term **barometric pressure.** Atmospheric pressure is usually stated in pounds per square inch, while barometric pressure is stated in inches of mercury. Thus, at sea level, the atmospheric pressure under normal conditions of the atmosphere is 14.7 pounds per square inch, while the barometric pressure at the same level is 30 inches of mercury column, or simply 30 inches.

TABLE I
WEIGHT OF DRY AIR

Temperature Degrees F. t	Weight of 1 Cubic Foot of Dry Air, in Pounds Avoirdupois			
	Barometer $B = 27$ Inches	Barometer $B = 28$ Inches	Barometer $B = 29$ Inches	Barometer $B = 30$ Inches
0	.07796	.08085	.08373	.08662
5	.07718	.08002	.08285	.08569
10	.07631	.07914	.08196	.08478
15	.07550	.07830	.08109	.08388
20	.07470	.07747	.08023	.08300
25	.07393	.07667	.07941	.08215
30	.07318	.07589	.07860	.08131
32	.07288	.07558	.07828	.08073
35	.07244	.07512	.07780	.08048
40	.07171	.07435	.07701	.07967
45	.07099	.07362	.07625	.07888
50	.07031	.07291	.07551	.07811
55	.06960	.07219	.07477	.07735
60	.06895	.07150	.07405	.07660
65	.06828	.07081	.07324	.07587
70	.06766	.07016	.07266	.07516
75	.06701	.06949	.07197	.07445
80	.06648	.06884	.07130	.07376
85	.06576	.06820	.07064	.07308
90	.06519	.06760	.07001	.07242
95	.06490	.06699	.06938	.07177
100	.06401	.06638	.06875	.07112
110	.06288	.06521	.06754	.06987
120	.06180	.06409	.06638	.06867
130	.06075	.06300	.06525	.06750
140	.05974	.06195	.06416	.06637
150	.05874	.06092	.06310	.06528
160	.05781	.05995	.06209	.06423
170	.05688	.05899	.06110	.06321
180	.05601	.05808	.06015	.06222
190	.05514	.05718	.05922	.06126
200	.05430	.05631	.05832	.06033
220	.05271	.05466	.05661	.05856
240	.05119	.05309	.05498	.05688
260	.04978	.05162	.05346	.05530
280	.04840	.05020	.05200	.05380
300	.04714	.04888	.05063	.05238
350	.04423	.04587	.04751	.04915
400	.04166	.04321	.04475	.04629

The weight of 1 cubic foot of dry air at any temperature and barometric pressure is found by means of the formula

$$w = \frac{1.3253 \times B}{460 + t}$$

in which w = weight of 1 cubic foot of dry air;
B = barometric pressure, in inches of mercury;
t = temperature, in degrees Fahrenheit.

The constant 1.3253 is the weight, in pounds avoirdupois, of 1 cubic foot of dry air at an absolute temperature of 1° F. and 1 inch barometric pressure.

EXAMPLE.—Find the weight of 1 cubic foot of dry air at a temperature of 60° F. and a barometric pressure of 30 inches.

SOLUTION.—

$$w = \frac{1.3253 \times 30}{460 + 60} = .0764 \text{ lb. Ans.}$$

Table I gives the weight of 1 cubic foot of dry air at different temperatures and barometric pressures, as calculated by the formula $w = \frac{1.3253 \times B}{459 + t}$.

83. Air, or Motive, Columns.—To illustrate, graphically, the movement of air through mines, Fig. 65 is taken and the assumption is made that the air in the mine is warmer than outside. Assume a to be the intake airway and b the upcast, or outlet, airway; the air will circulate from a, through c, and out at b. Suppose the air in the mine to be 75° F. with the barometer standing at 30 inches, while with the same barometer reading the air outside the mine has a temperature of 40° F. From Table I the difference in the weight of 1 cubic foot of air at 40° F. and 75° F. is $.07967 - .07445 = .00522$ pound, so that if the shafts were 200 feet deep there would be a pressure of $.00522 \times 200 = 1.044$ pounds per square foot more from the weight of the column of air in shaft a than in shaft b. This is termed the **motive column**, and has to overcome friction due to the air rubbing against the walls of the excavation in order that a circulation may take place.

Taking Fig. 65 again as an illustration, if the air outside the mine is warmer than that inside, the ventilation will be

stagnant, as no air would move. This, however, would not be the case if one shaft were longer than the other, for which reason a wooden chimney is constructed over one shaft. If the last expedient is not effective, artificial means should be adopted to create a circulation of fresh air.

Fig. 65

Friction has considerable bearing on the movements of air in mines; for instance, it increases as the square of the velocity and the length of the rubbing surface, and is inversely proportional to the sectional area of the passage through which the air moves. The deductions from these statements are that the areas of airways influence the circulation and that the length of airways influences the circulation; in fact, the flow of air through mines follows laws somewhat similar to the flow of water through pipes. To increase the quantity of air going through an airway of a given area, the power must be largely increased to gain the velocity necessary and overcome the friction, which increases as the square of the velocity. It is, therefore, a good plan to have main airways as large as possible and split the air; that is, carry it from the main intake airway to more than one level to supply the wants of the men and not attempt to carry the air in a continuous circuit so as to have the same air reach each working place in rotation, as is so frequently done.

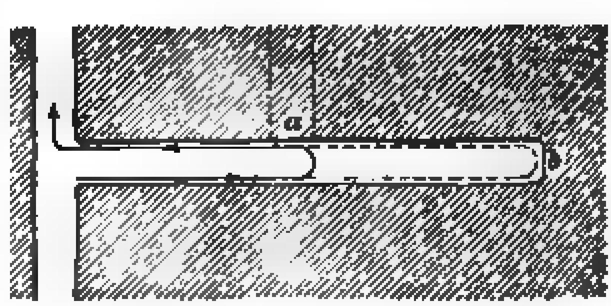


Fig. 66

84. Air on Levels.—Let Fig. 66 represent a level driven from a shaft. Since there is no opportunity for the heated air to rise and move away, it is evident that by diffusion and convection the air from the shaft will soon reach a point *a* that limits its circulation, as shown. In

order to induce the air to move to the face *b*, an artificial air-current must be established. This might be done by driving a winze, as shown by the dotted lines at *a* and then the air would take the direction shown by the dotted lines to *b*.

When the working place is a raise, Fig. 67, it is evident that the foul warm gases will remain at the top of the winze and that the nature of the excavation prevents things from improving, so that artificial means must be adopted if the men are to work, since diffusion and convection are very slow to act in such instances.

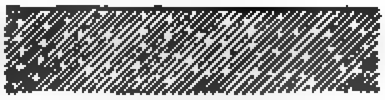


FIG. 67

85. Simple Tests for Circulation.—The direction of the air-currents not otherwise perceptible in horizontal workings may be ascertained by observing the flame of a candle. In a drift, the candle should first be placed on the floor, to test the lower current, and then fixed near the roof, to test the upper current. The direction of the air-current is indicated by the way the flame bends. The velocity of the current can be found by burning a little powder at one point; a second observer at, say, 100 feet away observes the time required for the smoke to reach him. The time required for fumes from a blast to reach a given point may also be noted.

86. Unconnected Workings.—It might be thought that there is no chance for ventilation in the case of unconnected workings; but so small are the differences required to set up a current that wherever work is going on the air is not absolutely dead. At the heading of a tunnel, air is heated by the burning lights and the animal heat of the men. This air rises to the roof, drawing in cooler air at the bottom to replace it; and if the tunnel is not too long, there will be a gentle outward flow along the roof to the mouth of the tunnel and an inward flow along the floor. This, for a certain distance, may suffice. When, however, the up grade of a tunnel places the heading too far above the mouth, this movement is stopped and the air at the face becomes permanently bad,

requiring artificial ventilation. The movement of air is down the sides of the shaft into the mine and upwards through the central portion of the shaft; but for extended workings this is greatly increased by dividing the shaft into compartments, one compartment being a downcast and the other an upcast.

87. Tunnel Connected With a Shaft.—Fig. 68 illustrates a shaft connected with a tunnel. If the temperature of the mine is above that of the atmosphere, the air will have a tendency to flow from *a* through the tunnel to *b*, and becoming heated, rise from the shaft from *b* to *c*. When the temperature of the atmosphere is above that of the mine, the current will be reversed and will flow from *c* down to *b* and out at *a*. The direction of the current would have been

FIG. 68

the same if in place of the tunnel there had been a shaft at *a* connected with the shaft *c b* by a drift at some distance below the surface. In other words, if there were no other factors entering into the problem, when the temperature of the mine is above that of the atmosphere, the current will always flow in at the lowest point and out at the highest, and a reversal of the conditions as to temperature will reverse the current. Unfortunately, there are many other factors that enter into the problem, and an engineer who has planned the connections with a view of having the shaft through which the men pass in and out the upcast and the other shaft the downcast may be disappointed by finding the conditions reversed. This does not argue ignorance of the laws of physics, or signify that there is any mystery about the principles, but only that, in a problem of much delicacy, the

necessary data are not always obtainable with precision. This difficulty, instead of discouraging the projector, should stimulate him to make the most careful observations and inferences before planning new connections for ventilation. In making plans, it is always best to have the men use the upcast for the passageway, and if possible to place the steam pipes, etc. in that shaft or compartment. The miner cannot come or go in the dark and if he moves against the current he inhales considerable lamp smoke, which is not beneficial; and when he stops for breath he is in a cool draft and liable to other dangers. On the other hand, it may be said that cool fresh air will not exhaust him as much as warm bad air that has been through the mine; this is, in a measure, true, but the lamp smoke is more injurious than the foul air, and this added to the chilling effect of the cool air probably outweighs any other consideration. When men are hoisted through an inclined shaft in special carriages, the hoisting shaft must be an upcast during the winter, for were it a downcast icicles might form and by falling injure the men. On this account, some mines that use this method of hoisting make provision for the building of fires in the upper levels near the top of the shaft through which the men are hoisted, and by this means warm the air and thus make sure that the current will always be an ascending one.

88. Disturbing Influences.—The wind may exert a disturbing influence on the direction of the ventilating current; for instance, a high wind striking in at the mouth of a tunnel or deflected down a shaft by a hillside or by buildings may cause the air-current to be reversed. The heat from the underground engines, steam pipes, etc. causes local disturbances of temperature, which generally assist ventilation, though in rare cases may retard it. The movement of cages, cars, pump rods, balance bobs, etc., and of rock in chutes, also, has, on the whole, a beneficial effect, though a stationary cage or car may temporarily block the circulation.

89. Sollars.—There are several ways in which natural ventilation may be accelerated and properly distributed

without much expense or trouble. The most common of these are for governing the currents.

Horizontal partitions in drifts or along main galleries by which the heated air from the working face is led out to the tunnel mouth or discharged into an upcast shaft are termed **sollars.** They are made of lagging or boards and are placed rather close to the roof. They assist the air movement, but have great disadvantages, and hence are seldom used in ore mines. Some of the disadvantages are as follows:

1. They require additional headroom, which means the excavation of a drift or a gallery considerably higher than otherwise needed, together with the extra expense for labor, explosives, and additional length of props.

Fig. 69

2. The introduction of any woodwork other than that actually needed, and especially woodwork of such light and inflammable character and in such an exposed position, is objectionable on account of the danger of fire and its cost. The wood has every opportunity for drying out and becoming ignited on small provocation. Fig. 69 illustrates a drift having a sollar formed by putting braces *b* below the regular caps *c* and placing a floor on top of the braces, thus providing the space *a*, which is the sollar through which the heated air from the mine flows out.

Sollars are also constructed along the floor of the drift or gallery. If constructed for the sake of ventilation, these are as objectionable as the others, but when constructed in connection with the water drain, they may be tolerated because

they are necessary. In such a case, however, the current of air passing in must oppose the current of water passing out.

90. Brattices.—**Brattices** are vertical partitions extending along and near one side of a gallery or drift. (If the passage is wide, the brattice may be in the center.) They are made of boards or cloth, which is usually tarred to prevent its rotting. Brattice cloths are considered objectionable, being a dangerous fire risk, and consequently do not find favor in metal mines. This objection is not reasonable, for fire risks may be minimized, and could be abolished entirely by the use of prepared cloth; the chief objection is probably that to purchase and hang the cloth would require expense. It should be more frequently used, as it will assist ventilation and thus be found economical, as better work will be accomplished by the men.

91. Wooden Air Boxes.—**Wooden air boxes** are sometimes employed and are much better than brattices or sollars. They are roughly made and do not have to be absolutely air-tight. They are usually placed in the upper corner of a drift or in the water drain. Their use adds another fire risk, but not to such an extent as when wooden brattices are employed. If used for return air, the end opening into the shaft should be extended upwards, so as to give a slight difference in elevation between the two motive columns; otherwise, they may not prove serviceable. In some instances, natural ventilation cannot be accomplished by air boxes; in which cases, hand fans, trompes, or other devices must be employed.

92. Metal Piping.—Pipes of thin metal (either tin or galvanized iron) are sometimes used as air conductors into working places in mines. They can be rectangular in section but are usually round. The same conditions apply to their use as were mentioned for wooden boxes.

93. Large Cloth Hose.—Ventilation is sometimes accomplished by the use of hose made from canvas or other fabric, but with natural ventilation the cross-section has to

be made so large that they are not much used. They are to be recommended for raises as they can be readily raised and lowered when necessary to clean out smoke or afford relief to men in the top of the raise. When attached to a small blower, their efficacy is most marked.

94. Air Doors.—**Air doors** are not used in metal mines to such an extent as at collieries, but when a mine has two shafts connected by a number of levels, air doors are necessary to prevent the air-currents from cross-cutting through the upper levels, and so leaving the lower workings unventilated. Air doors may be made of boards or plank, fitting closely to a frame or timber support, and they should always be self-closing. A simple arrangement is to hang a curtain of canvas across the gallery; with a good air-current this is not satisfactory and on levels it is a nuisance.

95. Lined-Shaft Compartments.—When men are hoisted on cages, it is considered safer to line the shaft compartments with boards or planks, which are usually set vertically with butt joists; but when the hoisting shaft is also used for ventilating, lined compartments have decided advantages. In cases of cage accidents, or when it is desired to pass from the cage compartment to another portion of the shaft, as to the water column (for inspection of repair work) it is more convenient to have no lining inside of the shaft timbers, and this is the usual custom in ore mines. When there are several compartments for hoisting, they are usually unlined. The compartment containing the water column, steam pipes, etc. is frequently lined and used as an upcast, the air passing down through the hoisting compartments. When this is done, air doors would be needed at the stations and especially at the bottom level.

96. Chimneys.—If the upcast does not draw well enough to furnish a good supply of air, a chimney built over it will correct the fault. This may, however, interfere with other arrangements, and then a chimney should be built over a shaft connecting with the first level; or a level may be run purposely to connect with the shaft to the mine, and

with a shallow shaft sunk for ventilation. In such cases, the ventilation may be very much increased by having a fire in the chimney. The height of the chimney need not be great, but it should be sufficiently high to afford a motive column. As the works extend, the height of the chimney may be increased, though it is better to build a fire in them.

97. Housed Shafts.—When a shaft has a house built over it to protect the men and the machinery from severe winter weather, the mine ventilation should not be overlooked. If the shaft is an upcast, there should be a hood, belfry, or cupola-like structure in the top of the shaft house to allow free escape of the mine air and vapor; if it is a downcast, the air admitted should be fresh and free from smoke or dust. Shafts with houses erected over them are not recommended on account of dangers arising from fire. Their only excusable use is during the temporary work of sinking new shafts.

98. Wind Sails.—Fresh air may be forced to the bottom of a shaft of moderate depth by setting up funnel-shaped ventilators that can be turned to face the wind, and connecting them with the shaft bottom by means of pipes or large canvas hose. This is a simple and convenient makeshift to use while sinking uncovered shafts 100 or 200 feet deep.

99. Effects of Water on Ventilation.—When a mine is wet, the water may assist in its ventilation. For instance, if water is dripping down a shaft, the upcast may be lined and the water kept from passing down through it. The water dripping down the other compartments of the shaft will have a tendency to carry air with it, thus creating a current down the hoisting compartments and up through the compartment that is lined. When sinking in advance of regular mine work, whatever water has a tendency to flow down through the shaft may be confined in a wooden box or launder; and if this is made of sufficient size the current of water will carry air with it, thus ventilating the bottom of the shaft. If water is allowed to flow from one level of a mine

to another, the downward flow can be confined to the winzes or shafts that are intended as downcasts, the upward current of air passing through other openings. In this way, the falling water can be made to assist in the ventilation of a mine.

100. Advantages and Disadvantages of Natural Ventilation.—Wherever it is practicable to get along without the use of blowing or suction machinery, the ore miner will inevitably do so. Natural ventilation has the advantage that it costs nothing after the connections are once made and it takes care of itself for the most part. But it is often insufficient; it is not always reliable, fluctuating with the weather, the time of day, the wind, and artificial disturbing causes; and while it costs nothing for maintenance, it may require a considerable initial outlay in making connections or sinking (or raising) air-shafts that would not otherwise be needed. As against this latter point, it may be remarked that most of the work in developing fits in with that done to gain air connections and vice versa.

All ore mines are started on the basis of natural ventilation. When that becomes unsatisfactory, the various assisting expedients come into play.

ARTIFICIAL MEANS OF VENTILATION

101. Trompe.—When natural draft (assisted by simple means) is insufficient, metal mines require artificial ventilation. In some regions, ventilation has been secured by the use of a **trompe**, or waterfall ventilator. This is illustrated by Fig. 70, in which *W* represents the water flowing through the trough *L* on to the screen *G*. This screen breaks the water into a shower of drops or fine streams that fall on the dashboards *D*. This shower of water being deflected from one dashboard to another becomes beaten into a spray, and in its descent draws air through the openings *A*. After leaving the last dashboard, the water falls through the remainder of the tube to the trap *W T*, from which it overflows and escapes. During this long fall, the water acquires a considerable velocity, causes a vacuum behind it, and compresses

the air before it, thus causing rapid currents in through the openings *A* and out through the discharge pipe *V*. While the trompe is not used in many American ore mines, the principle is of great assistance in the ventilation of mines; and by causing the mine waters (on their way to the pump levels) to pass through certain compartments in the shafts or through certain small shafts, these can be made downcasts; by keeping the water from passing through similar compartments or shafts, they will become the upcasts.

102. Furnaces are not much used at metal mines as they are dangerous with one shaft even where little timber is used. When there are two shafts, however, they may be used at the top of lined shafts, or at the bottom of unlined shafts without danger and with excellent results.

103. Machine Ventilators. Machines used for ventilating ore mines may be divided into three classes: *compressors*, *blowers*, and *fans*.

1. *Compressors.*—Air compressors are never installed for ventilating purposes only, but when used for driving air drills or other underground machinery, they furnish a large amount of air to the workings, and may thus be considered as ventilating machines. Then, again, when air pipes are in use in a mine, branches may be taken from these pipes to ventilate certain sections or portions where the drills or other machinery driven by compressed air are not in use. Although the air pipe is small, it carries a great deal of air (measured by its expanding volume at atmospheric

Fig. 70

pressure). The delivery at the drills is usually about 70 pounds per square inch; and when this air is released it expands and becomes cooler, thus reducing the temperature at the working face. At times, when the drills are not running, a small amount of air may be allowed to pass through the air hose, thus furnishing the men with the necessary ventilation; and when powder smoke is to be removed, the full head of air may be turned on for a few moments.

2. *Blowers.*—The positive rotary blowers take in and discharge a given amount of air at each revolution; Fig. 71 illustrates one of this type. *A* is the pulley by means of which the machine is driven; *B* and *C* are cases containing gears that drive the upper blower shaft from the lower one; *a*, *b* are the impellers, or veins, that rotate about the shafts *c*, *d*

FIG 71

in such a manner as to draw the air in at *g* and drive it out at *f*, as shown by the arrows; *e* is the cast-iron case surrounding the impellers. The impellers do not come in contact with each other or with the case, but are simply a close working fit, the two shafts being kept in the proper relation by means of the gears in the cases *B* and *C*. The arrow *V* shows the direction of rotation of the shafts and also the direction of the air-current. This style of machine is especially useful where it is desired to furnish a given amount of air at a certain point in the workings (usually a long distance from the blower). Blowers of this class have been used for ventilation when driving tunnels. In one case, the air pipe was carried into the face of the tunnel and the blower could be operated either to drive fresh air in or to

draw out the gases resulting from blasting. Positive blowers can force a large amount of air through a comparatively small pipe on account of the pressure they can produce.

3. *Fans*.—American practice in the mechanical ventilation of mines varies widely from English and Continental methods. Abroad, they adhere to the large low-pressure centrifugal fans, of which the Guibal is a well-known type. These large fans are well suited to collieries, as they handle great volumes of air at low pressure. They are usually run as exhaust fans and are placed at the side of a shaft mouth, from which they suck out the air. Some fans of this class are as much as 50 feet in diameter, and the medium sizes would be regarded as clumsy and unnecessarily large at most ore mines. They are run at relatively low speeds. A few of these large fans are seen at American ore mines, but preference is given to the small high-speed fans. Fans are used more frequently than blowers, and some of them develop pressures or vacuums comparable with blowers, and hence are effective for the longest distances reached by mining.

As to mode of action, there are two main divisions of fans—centrifugal and propeller; the former class is the more important. A simple apparatus like the paddle wheel of a side-wheel steamboat on being rapidly rotated will produce a tangential movement of the air by centrifugal force, the outgoing air being replaced by air drawn in at the center. This principle is taken advantage of by a multitude of devices giving improved efficiency over that obtained by merely beating the air with plain flat radial blades or vanes.

Fig. 72 illustrates a small blower fan having an electric

motor placed inside the fan. Machines of this style are especially adapted for working in comparatively close quarters and for delivering air to some portion of a mine that could not be reached by natural ventilation.

104. The Ventilation of Isolated Places.—For furnishing air to workmen at the tops of raises, or in rooms that the ordinary ventilation cannot reach, small fans are sometimes placed at various points in the mine, which take air from the intake of the mine and force it to these unventilated workings. These fans may be driven by hand or are made to run by small waterwheels driven by the mine waters as they pass from one level to another. Fans driven by electric motors may also be used for this purpose. Some mines that are working deposits extending over considerable territory in one direction (as the drift gravel mines of California) employ blowers that force the air through the tunnel and to the working face. When more than one working face is being operated, the air pipe may be branched and part of the supply carried to each face; by this means, the workmen are always supplied with good fresh air.

MINE SANITATION

UNDERGROUND SANITARY PRECAUTIONS

105. Air.—The most important factor in mine sanitation is the **air** that the men breathe. The methods of securing pure air have been considered.

106. Water.—Next in importance after pure air is the **water** that the miner drinks. In many mines, the natural waters flowing into the mine are unfit for drinking, either on account of the chemicals that they contain or because they may be largely surface waters, carrying more or less sewerage from the mining location. In either case, the miner should never be allowed to drink the underground waters, but should be provided with fresh and pure drinking water.

107. Powder Smoke.—**Powder smoke** is also an important factor in mine sanitation. The gases produced by some forms of high explosive are very injurious to the health of the miners. On this account, the mining authorities should procure the least objectionable explosive that will suit the conditions occurring in the mine. This precaution is especially necessary when men are forced to carry on work where there is no natural or artificial ventilation, as at the head of a raise or at the end of a prospecting tunnel.

108. Animal Filth.—The presence of **animal filth** in the working should never be allowed under any conditions, and sinks or special provisions should be made for it. These should be placed in the upcast, or return airway, and should be so constructed that the tubs can be brought to the surface and cleaned every few days. This is one of the most important points in regard to mine sanitation, and should never be neglected.

SURFACE PRECAUTIONS

109. Change Houses.—After the miners come from work in wet mines, they should always change their wet clothes and take a bath. For this purpose, the mine should provide a dry, or change, house. This may be a separate building, provision being made for drying the clothes either by steam heat or by stoves. The change house and its baths should always be kept in good order and as clean as possible. Lockers should be provided for the men to keep their surface clothes in, and an attendant should always be in charge of the house. This change house should never be allowed to become a loafing or smoking place, or contain vermin.

110. Sanitary Arrangements at the Mine Location. If the mine owns the houses in which the men live, these should be so placed that they will have a good natural drainage; and if this is not the case, sewers or drains should be provided. Care should be taken in regard to the water

supply for the mine location, as imperfect drainage or an impure water supply may bring sickness and disease among the miners, which would more than counterbalance the expense of providing good surface arrangements. Mountain fever is a species of typhoid fever nearly always induced by impure drinking water.

111. Remarks.—Though the miner's work is largely of such a nature as to undermine his health, there have been precautions advocated that will prevent sickness and add to the length of his days also. The necessity of their taking a bath every time that they come from the workings has the effect of keeping the workmen clean, and undoubtedly is conducive to a better state of health. Most mines are in locations where the air is fresh and pure and the water supply good, and if the miner will exercise a little in the open air to remove the lamp smoke from his lungs, he undoubtedly will remain a strong healthy man.

SUPPORTING EXCAVATIONS

(PART 1)

TIMBER SUITABLE FOR MINING

STRUCTURE OF WOOD

1. Mine Timbering.—In the English language, there is no word that includes the various means adopted to support weak walls in mines. **Timbering** is not a precise term, since masonry, concrete, and iron are used successfully; besides, in some mines, timbers are used for other purposes than supporting excavations. But for lack of a better expression, all may be placed under the general heading **supporting excavations**, and subdivided for itemized description. The pressure that acts on the rocks in which the excavation is made may come on any one or all four of the walls to such an extent as to cause them to break and fill the excavation. In bed mining, the top wall, termed the *roof*, is the one that needs the most support, although the side walls, termed *ribs*, or the bottom wall, termed the *floor*, may need support.

The use of timber in mines is not confined to supporting excavations, but is used in directing the air-current circulating in the mine so that proper ventilation will be given the miners. Again, wood is used for chutes, platforms, gates, and batteries for the purpose of handling ore after it is broken. It is also used for track cross-ties, for covering sumps, for underground stables, engine rooms, fencing off dangerous places, and for other purposes either of a temporary or a permanent nature.

2. Timbers Used in Mines.—The trees that grow near mines are the ones generally used for mine timbering. If the wood from such trees is of inferior quality and durability, it is only valuable for temporary purposes where strength is of secondary importance; on the other hand, if the wood is strong and durable, it is used where permanency is desirable. It is a mistaken idea that any wood is suitable for mine timbering, for often it will be found cheaper in the end to purchase expensive timber, than to use cheap wood that must soon be replaced. It is also a mistake to assume that durable wood will answer in every situation underground, for it will not, since it will be attacked and quickly destroyed by disease if not properly dressed and cared for. There are two extremes where timbering fails—one where weak brash wood is used, the other where strong durable wood is allowed to become diseased. In the first case, economy is a failure; in the second case, negligence is the cause of economical practice becoming a failure; both examples show, therefore, the necessity for understanding the structure of wood, the particular purposes for which it is adapted, its durability, physical properties, diseases by which it may be destroyed, and the methods that may be adopted for its preservation.

3. Classes of Trees.—There are two classes of trees suitable for mine timbers—one known as **coniferous,** or **evergreen, trees,** and the other known as **deciduous trees,** or those whose leaves fall off in autumn. Both classes of trees are exogenous, that is, they add each year a new layer of wood, which covers the growth of the previous year. The situation in which a tree grows has much to do with its structure; for instance, trees that grow in swampy ground have larger pores than those that grow on uplands. Structure has much to do with the strength of wood and, as a rule, a close-grained heavy wood is stronger than one having an open porous structure. Conifers, or needle-leaved trees bearing cones, and deciduous, or broad-leaved, trees differ in their structure and character, the wood of the former being

called soft and that of the latter hard wood, although some conifers are harder than the so-called hardwoods. The wood fibers of conifers, composing the main part of the wood are all alike and their arrangement is regular. The wood of the broad-leaved trees is complex in structure and lacks the regularity of arrangement so noticeable in conifers.

4. Annual Rings.—The wood next to the bark being of more recent growth is porous, compared with that nearer the center of the tree. The **sap wood** is the outer portion of a tree and the **heart wood** is the inner portion. In the sap wood, cells for the process of growth and the running of sap are more pronounced than in the heart wood, the latter not being so active in the growth of the tree. Each year's growth is marked by an annual ring; and by counting these rings, it is possible to determine the age of the tree. The rings vary in width for the same kinds of trees according to their quick or slow growth, and also vary in width according to the kind of tree. The rings vary in thickness from $\frac{1}{50}$ inch in less thrifty trees to $\frac{1}{8}$ inch in very thrifty trees; they probably average $\frac{1}{12}$ inch for thrifty trees. Generally, the rings nearest the center of the tree are the widest. In most trees, each ring is made up of an inner, softer, light-colored part and an outer, firmer, darker-colored portion. The light-colored part is termed *spring wood*, and the outer darker portion the *summer wood* of the ring.

5. Comparison of Pine and Oak.—Fig. 1 shows a plank cut lengthwise from a pine log. The annual rings of light spring wood and of dark summer wood are conspicuous on the cross-section *a*; they are also shown on the tangential section *d*, where *b* represents the spring wood and *c* the summer wood. The annual rings show as straight lines on the radial section *e*

FIG. 1

Fig. 2 shows a plank cut lengthwise from an oak log. The annual rings *a* show on the cross-section and also the medullary rays *e*, which cross the annual rings in radiating from the center of the tree. On the tangential face *d*, the medullary rays show as long tapering lines *e*; while on the radial face *b*, they show as broad bands *e*. In oak, the spring wood is darker than the summer wood.

Fig. 2

6. Seasoning Wood.—When timber loses its water, it is said to be seasoned, and weighs less than in the green state and at the same time it becomes more durable. Timber, when dried quickly, has a tendency to absorb moisture when again exposed, and also to split, or *check*, along the grain. Mine timber should be well seasoned by air drying; and, since the ends of timber will dry first, they will check, but this is local, and sometimes disappears as seasoning progresses. From the time timber is felled until placed in the mines, it should be watched carefully to prevent it from becoming diseased. If timber is left many months in water, it becomes water logged and sinks; at the same time, some of the soluble materials in the wood are leached out without much impairing its strength. For some purposes, the seasoning of wood is hastened by artificial processes, the most common of which is drying the wood by heat in a kiln.

7. Weight of Wood.—Green timber contains much sap and moisture and weighs more than seasoned or dried wood.

Seasoned wood is heavier at the center and green wood is heavier on the outside. Butt wood, or that nearest the ground, is usually heavier than the top wood of the same tree. In general, native woods when seasoned are not so heavy as water; in fact, few weigh as much as 50 pounds per cubic foot, and some of the pines and conifers weigh less than 30 pounds per cubic foot.

TABLE I

APPROXIMATE WEIGHT OF DRIED WOODS

	Specific Gravity	Weight, in Pounds	
		Per Cubic Foot	Per 1,000 Feet Board Measure
Very Heavy Woods.—Hickory, oak, persimmon, beech, locust	.7 to .8	43–50	3,900
Heavy Woods.—Ash, elm, cherry, birch, maple, long-leaved pine, and tamarack	.6 to .7	37–43	3,300
Medium Weight Woods.—Pitch pine, Douglas spruce, Western hemlock	.5 to .6	31–37	2,800
Light Woods.—Norway and bull pine, red cedar, cypress, hemlock, spruce, fir, redwood, basswood, butternut, tulip, buckeye, yellow poplar . . .	.4 to .5	25–31	2,300
Very Light Woods.—White pine, spruce, fir, white cedar, poplar	.3 to .4	19–25	1,800

8. Felling Trees.—It has been customary to fell timber in winter in the belief that in this season trees do not contain so much water as in other seasons, but this idea has been refuted and the understanding is now that the trees contain as much water in winter as in summer, but not so much sap. As soon as trees are felled, their bark should be removed to prevent insects from injuring the wood and to prevent the sap from fermenting during seasoning and causing decay. The timbers should not be permitted to

lie on the ground after seasoning operations are commenced but should be placed on blocks so that they will be exposed to a circulation of air. The blocks should not be so far apart that the timbers will sag, and the timbers, if exposed to the sun, should be turned regularly, otherwise they may check and *warp*. Sawed timbers should be stacked up with air spaces between the sticks; they should also be kept under sheds when seasoning and before they are taken below ground. If this is not possible, they should be stacked so that they will shed water. These precautions may seem unnecessary to some, but if carried out will prove beneficial to the timber and economical to the mine management.

9. Shrinkage of Wood.—One of the many peculiar features in the physics of timber is that wood does not shrink appreciably in the direction of the length of its fibers, but does shrink at right angles to them. This may cause checks in the direction of the length of the wood fibers, particularly if seasoning is carried on unevenly or so quickly that the stick does not lose its moisture uniformly. If the moisture dries out on one side of a stick more rapidly than on another, the stick will twist and warp.

Fig. 3 shows the effect of shrinkage; it will be noticed that the checks are lengthwise of the wood, for the reason

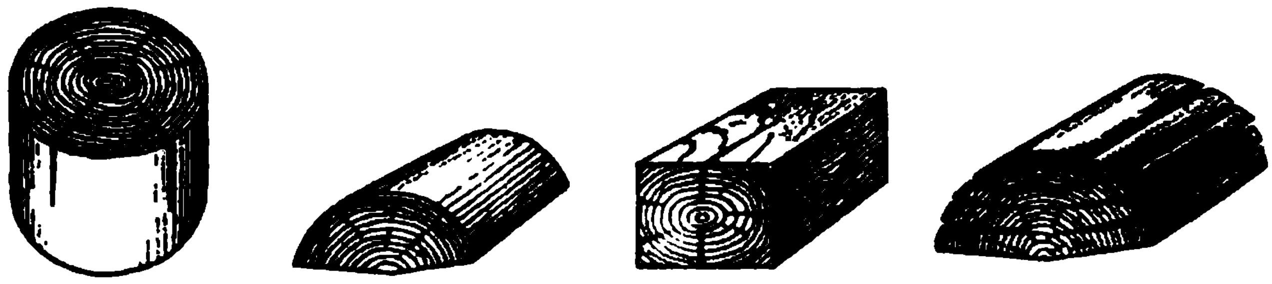


FIG. 3

that longitudinal shrinkage is very little, while shrinkage across the grain is considerable. One of the great troubles in wood seasoning is the difference between the amount of shrinkage along the radius and that along the tangents to the rings. Sap wood, as a rule, shrinks more than heart wood; and the hardwoods shrink more unevenly and check to a greater extent than the conifers, which phenomenon is probably due to the regular structure of the latter.

10. Selection of Mine Timbers.—It is poor policy to select poor mine timbers where durable timber may be obtained for a little more money, since for permanent purposes in mines the best wood is none too good; and if one uses hemlock where oak, long-leaved pine, or redwood can be obtained, it is practicing false economy. It is not necessary to select mine timbers as woodworkers select lumber, yet sound sticks free from *shakes* and, as far as possible, checks should be chosen and inferior sticks rejected, particularly if the timbers must be purchased. In the selection of timbers for mining purposes, their strength and enduring qualities are to be considered. Timbers used as props or posts have to resist crushing endwise; timbers acting as beams must resist bending, hence, they must possess stiffness, for while the upper side of such a beam is in compression across the grain the under side is in tension.

11. Shakes are splits in a log. When trees are subjected to high winds, the cross-fibers that bind the longitudinal fibers are sometimes broken and by this means a series of tubes are formed composed of annual rings one inside the other. Such trees are said to be **wind shaken** and make poor lumber. Wood is said to be **heart shaken** when it is checked across its center, or heart wood; when it is checked radially, it is said to be **star shaken.**

12. Durability of Mine Timber.—All wood is equally durable under certain conditions. Kept dry or submerged in water, it lasts indefinitely. Timbers in the Alten Mann mine in Saxony have been in place 300 years and are still in sound condition, but this is exceptional and is only to be accounted for by their being wetted daily. Where fresh dry air can circulate about timbers, they will last indefinitely in a dry place; under these conditions timbers in some mines in this country have been in place over 40 years. When wood is not properly seasoned, the sap is liable to ferment, especially in a dry warm place and **dry rot** occurs, beginning in the center of the stick and working outwards. In general appearance, such a stick looks sound; but by thrusting

a knife blade into it the damage is discovered. When the stick is away from decaying timber, a circulation of fresh air is one preventive of dry rot, as in such situations the stick seasons.

13. Damp Rot.—When timber is placed in warm moist air, **damp rot** takes place. This is the usual rot affecting mine timbers. It commences on the outside and gradually finds an entrance into the interior of the stick through some check. The destruction of a timber by damp rot is not so rapid as by dry rot and is noticeable from the fungus growth on the outside of the stick. In mines, dry rot occurs in the return airways and in poorly ventilated workings, while damp rot occurs in the intake airways and damp rooms. When fungus of the damp-rot species appears, it may be possible to save the timber and prevent the fungus from reaching the heart wood by washing the stick with lime or alum water from time to time.

14. Preservation of Timber.—The partial removal of sap will retard decay, for which reason timbers are sometimes submerged for several months, then removed and air dried. Warmth between 60° and 100° F., combined with moisture, is favorable to decay, but mine timbers must often be placed in such situations. It may be possible to increase the life of timbers by using special wood preservatives, but even then the sap must be either dried or removed, since wood covered with paint before it is thoroughly seasoned will propagate dry rot in a warm dry place, or damp rot in a moist warm place. Creosoted joints will prolong the life of good sound timber, especially if the ends of the timbers have been submerged in creosote a month or more. Different species of trees differ in their resistance to decay. Under conditions favorable to decay, cedar and locust are more durable than pine, oak, or cypress, although in certain situations they may all have the same durability. Contact with earth is particularly destructive to timber; and nearness to decaying timber is also another source of disease. Barked timber is the best for mines, since any timber that has been felled and permitted

to remain with its bark on, even a short time in summer, will attract insects and worms, and the bark will prevent its seasoning. Sap wood is more subject to decay than heart wood.

15. Practical Remarks.—In framing, where stiff timber is wanted, the conifers excel. Where heavy but steady loads are supported, the better and heavier conifers answer as well as hardwoods; on the other hand, where the load is movable and the timbers are subject to shocks and jars, the tougher hardwoods are safest.

In shaping wood, it is better to split than to saw it because splitting insures straight grain and secures a more nearly perfect seasoning. Checks, knots, and shakes are always a source of weakness. Rafted timber and artificially seasoned or kiln-dried timber are as durable as any. Summer-felled timber does not season so rapidly as winter-felled and offers greater inducements to insects and fungi to develop.

TIMBER MEASUREMENTS

16. Introduction.—The mine manager is usually able to purchase timbers and seldom has to bother with standing timber. An experienced woodsman can tell at a glance about what size stick can be cut from a standing tree; and by measuring the circumference with a tape and the height with his eye, he can tell almost exactly what size square stick a tree will furnish. In case the mine manager has a large timber tract and uses much timber, he should have a capable woodsman take charge of the timber, for he cannot well afford to occupy his mind with such matters.

The woodsman who does the felling should see that no good timbers are wasted and that timbers are cut to proper lengths. In case a small tree will answer the purpose, a large one should not be felled and its top taken, for the light wood from the top of an old tree is not so strong or so durable as wood from a younger tree.

17. Cost of Timber.—Timber varies greatly in cost, according to locality and the price of labor. For example,

sticks that cost 2 cents per running foot east of the Mississippi will cost 5 cents per running foot west of that river. It is usual to stipulate that a timber shall have a certain minimum diameter at its smaller end, and it is then purchased by the running foot.

A stick 8 to 12 inches in diameter at the small end costs on an average 5 cents per running foot in Colorado; then a stick 16 inches in diameter will cost 9 cents; one 20 inches in diameter, 14 cents; and one 24 inches in diameter, 20 cents per linear foot. These prices are based on the increased quantity of wood in a stick, with the area of a circle 12 inches in diameter as the base, and 5 cents the base price. If timbers 12 inches in diameter can be obtained cheaper than the price named, the others should be proportionally less in price.

18. Size of Mine Timber.—Sticks smaller than 8 inches in diameter are merely useful for temporary timbering or for lagging. As a usual thing, timbers for framing are from 8 to 12 inches on a side when squared, and probably timbers of the smaller size are the most used. A round timber used instead of a squared timber should be of such diameter that it will furnish approximately the same area as the sawed stick that it replaces. For example, if a 10-inch sawed stick is to be replaced by a round stick, the latter should be approximately 12 inches in diameter, according to the rule $d^2 \times .7854$ = area of a circle. On traveling ways, framed timbers are usually about 7 feet 2 inches high in the clear, but this may be varied by the management.

The caps are varied in length to suit conditions, but 8 feet is a fair length. Beams are used in various sizes and lengths, while props vary from 6 inches to 24 inches in diameter.

LOGS REDUCED TO SQUARE TIMBER

19. The Two-Thirds Rule.—In the **two-thirds rule** for determining the amount of square timber contained in logs, allowance is made for the waves that occur, logs being seldom perfectly round and straight. The diameter of the

log is found from the circumference taken at its middle length, or the diameters of the two ends of the log are added together and divided by 2 to obtain the average diameter. The diameter of the log is reduced one-third to allow for slab, and the remaining two-thirds is taken as the width of the square piece that may be hewn or sawed from the log. The cubic contents of the squared log are then obtained by squaring its width and multiplying by the length of the log.

EXAMPLE 1.—The circumference of a log at its middle length is 37.7 inches; according to the two-thirds rule: (*a*) what will be the area of the stick when squared? (*b*) what will be its contents, in cubic feet, when squared if the log is 10 feet long?

SOLUTION.—(*a*) Since 3.1416 × diameter = circumference, 37.7 ÷ 3.1416 = 12 in., or the diameter of the log. According to the rule, one edge of the stick is

$$\frac{12 \times 2}{3} = 8 \text{ in.;}$$

hence, the area is

$$8 \times 8 = 64 \text{ sq. in. Ans.}$$

(*b*) Since the log is 10 ft. long, or 120 in., it contains

$$\frac{64 \times 120}{1{,}728} = 4.44 \text{ cu. ft. Ans.}$$

Or

$$\frac{64}{144} \times 10 = 4.44 \text{ cu. ft. Ans.}$$

EXAMPLE 2.—The diameters of a log at its ends are 12 and 16 inches; calculate, by the two-thirds rule, the width of a square stick cut from this log.

SOLUTION.—The average diameter is

$$\frac{12 + 16}{2} = 14 \text{ in.}$$

Then, by the rule, the width of the stick is

$$\frac{14 \times 2}{3} = 9\tfrac{1}{3} \text{ in. Ans.}$$

20. The Inscribed-Square Rule.—The **inscribed-square rule** gives larger results than the two-thirds rule, since no allowances are made for imperfections in the log or for saw cuts. The exact mathematical rule for determining the side of a square inscribed in a circle is as follows: The side of the square is the hypotenuse of a right triangle two sides of which are radii r of the log. Hence, the side of

the square is $\sqrt{r^2 + r^2} = \sqrt{2r^2}$, or the square root of the sum of the squares of the other two sides. Since $r = \frac{d}{2}$, that is, the diameter is twice the radius,

$$r^2 = \frac{d^2}{4}, \text{ and } 2r^2 = \frac{d^2}{2}$$

Therefore, the side of a square inscribed in a circle equals

$$\sqrt{\frac{d^2}{2}} = \sqrt{.5d^2} = .7071d$$

The area of the inscribed square will be equal to the side squared or $2r^2$ or $\frac{d^2}{2}$.

EXAMPLE.—The circumference of a log at its middle length is 37.7 inches; according to the exact mathematical rule for determining the area of a square inscribed in a circle, what will be the area of the stick when squared?

SOLUTION.—Since 3.1416 × diameter = circumference, 37.7 ÷ 3.1416 = 12, or the average diameter of the log. Then, according to the rule

$$\sqrt{\frac{d^2}{2}} = \sqrt{\frac{144}{2}} = \sqrt{72} = 8.48 \text{ in.},$$

the side of the inscribed square, and 8.48 × 8.48 = 72, the area in square inches. Again, since the radius is one-half the diameter, the radius squared multiplied by 2 gives the area or 72 sq. in. Ans.

21. The 17-Inch Rule.—The **17-inch rule** is based on the fact that a log 17 inches in diameter will square 12 inches. The contents, in cubic feet, of a log are obtained by this rule as follows:

Rule.—*Multiply the square of the diameter of the log, in inches, by its length, in feet, and divide by the square of 17.*

The width of a square piece may be obtained by multiplying the diameter of the log, in inches, by 17, and dividing the result by 24, or by multiplying the diameter by .7083.

This rule for calculating the width of the inscribed square piece is based on the fact that one side of the square inscribed in a circle 24 inches in diameter is 17 inches long.

22. The Quarter-Girth Rule.—One method frequently adopted for calculating the cross-sectional area of the square

piece that can be furnished by a log of a given diameter is deduced from the following reasoning:

The circumference of the log is measured at its middle length with a tape, and equals $2\pi r$. If the side of the square were one-fourth of the circumference of the log, or $\frac{2\pi r}{4}$, the area would be $\left(\frac{2\pi r}{4}\right)^2$. But one-fifth must be deducted from this quantity to compensate for the loss in cutting; hence, the area of the square is $\frac{4}{5} \times \left(\frac{2\pi r}{4}\right)^2 = \frac{\pi^2 r^2}{5}$ $= 1.97\, r^2$. This gives almost exactly the area of a square piece that can be cut from a log. The bark should be removed from the log before it is measured.

Example.—The diameter of a stick is 12 inches at its middle length; what will be the area of a stick sawed from such a log, according to the quarter-girth rule?

Solution.—The radius of such a log is 6 in. and

$$1.97 \times 6^2 = 70.92 \text{ sq. in. area. Ans.}$$

TO COMPUTE THE VOLUME OF ROUND TIMBER

23. Molesworth's Rule.—The following rule may be used to compute the solid contents of a log:

Rule.—*Add together the squares of the diameters of the greater and lesser ends, and the product of the two diameters; multiply the sum by .7854 and this product by one-third the length of the log.*

Example 1.—The diameters of a log at its ends are 2 feet and 1.4 feet, and its length is 15 feet; what number of cubic feet does it contain?

Solution.— $2^2 + 1.4^2 + (2 \times 1.4) = 8.76$

and $8.76 \times .7854 \times \frac{15}{3} = 34.40$ cu. ft. Ans.

When the length is in feet and the diameters are in inches, proceed as in example 1 and divide the result by 144 to obtain the volume in cubic feet.

Example 2.—The diameters of a log at its ends are 24 and 18 inches and its length is 15 feet; what will be its volume in cubic feet?

SOLUTION.—
$[(24^2 + 18^2 + 24 \times 18) \times .7854 \times 5] \div 144 = 36.32$ cu. ft., volume. Ans.

When the dimensions are in inches, proceed according to the rule and divide the result by 1,728 to obtain the volume in cubic feet.

EXAMPLE 3.—The diameters of a log at its ends are 20 and 15 inches and its length is 144 inches; what will be its volume in cubic feet?

SOLUTION.—
$$\left[(20^2 + 15^2 + 20 \times 15) \times .7854 \times \frac{144}{3}\right] \div 1{,}728 = 20.18 \text{ cu. ft.} \quad \text{Ans.}$$

The strongest beam that can be cut out of a log is one in which the breadth is to the depth as 5 to 7 very nearly, and can be found graphically as shown in Fig. 4. Draw any diameter *a b* and divide it into three equal parts *b f*, *f d*, and *d a*. From *f* and *d*, draw lines perpendicular to the line *a b* to meet the circumference at the points *c* and *e* and connect the points *a*, *e*, *b*, *c*, and *a* as shown. If the log is 12 inches in diameter, *a e* will be nearly 7 inches and *a c* nearly 5 inches.

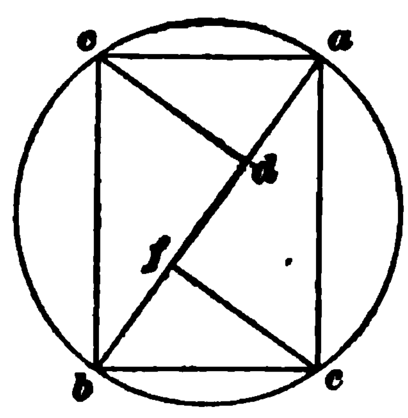


FIG. 4

24. Ordinary Rule.—For all practical mining purposes, the cubic feet in a log may be obtained by finding the area of its middle cross-section and multiplying this by its length. The area of its middle cross-section may be found from its perimeter at its middle length, or from an average diameter obtained by taking the diameter at each end and dividing their sum by 2. In case the length is in feet and the diameter in inches, divide by 144; and if all dimensions are in inches, divide by 1,728 to obtain the volume in cubic feet.

LUMBER MEASUREMENTS

25. Measurements of Squared Timbers.—The mine manager has much to do with squared lumber, and very often is obliged to reduce the number of cubic feet in a stick

to the number of feet, board measure. Cubic feet may be changed to board measure by multiplying by 12.

Rule.—*To find the number of cubic feet of squared timber when all the dimensions are in feet, multiply the breadth by the depth and that product by the length. If one of the dimensions is in inches, multiply as before and divide by 12. When any two of the dimensions are in inches, multiply as before and divide by 144.*

26. Board Measure.—Framing timber is usually sold by board measure.

Rule.—*To reduce lumber to board measure, multiply the length in feet by the width and thickness in inches, and divide the product by 12.*

A board 1 inch thick, 12 inches wide, and 12 feet long will contain 12 feet, board measure, which is found thus:

$$\frac{1 \text{ in.} \times 12 \text{ in.} \times 12 \text{ ft.}}{12} = 12 \text{ feet, board measure.}$$

A board 6 inches wide and 10 feet long will contain

$$\frac{1 \text{ in.} \times 6 \text{ in.} \times 10 \text{ ft.}}{12} = 5 \text{ feet, board measure.}$$

Planks are more than 1 inch thick and are sold by board measure; thus, a plank $1\frac{1}{2}$ inches thick, 10 inches wide, and 12 feet long would contain

$$\frac{1\frac{1}{2} \text{ in.} \times 10 \text{ in.} \times 12 \text{ ft.}}{12} = 15 \text{ feet, board measure.}$$

EXAMPLE.—How many feet, board measure, are there in sixteen joists 3 in. × 4 in. and 16 feet long?

SOLUTION.—

$$\frac{16 \times 3 \text{ in.} \times 4 \text{ in.} \times 16 \text{ ft.}}{12} = 256 \text{ ft., board measure. Ans.}$$

In the specifications for buildings and the purchase of lumber, board measure is of importance in determining the cost. Planks for lagging or caps are also purchased by board measure.

MECHANICS OF TIMBERING

27. Strength of Timber.—All calculations for determining the strength of timber consider the material as a bundle of parallel fibers; such fibers, according to the nature of the wood, have a definite strength, which is determined by experiment. As the experiments are made with sound wood under conditions that are not likely to exist in practical work, a factor of safety is used; that is, the experimental strength is divided by any factor 3, 4, or 6, etc. that is thought advisable. Timbers may be subjected to loads that act mostly in a direction parallel to their fibers or in a direction across their fibers. A post that supports a load is under compression, which tends to shorten the post. The fibers of wood act as little columns firmly grown together; and when posts break by compression it is due to fibers separating into a large number of small independent pieces. Like the strands of a rope, the independent pieces of fiber offer little resistance to compression and they bend over and the piece buckles.

BENDING

28. Compression and Elongation.—When a stick bears a load either parallel with or across its fibers, it is subjected both to compressive and to tensile stresses. The fibers of a beam supporting a transverse load have a tendency to bend on the lower side; and in case the beam does bend, the fibers are compressed on the upper side. This is shown in Fig. 5, where the fibers are torn apart on the under side of the beam and pushed together on the upper side. The fiber at or near the center of the beam is neither lengthened nor shortened and is called the **neutral axis.** If the ruptured fibers pass the neutral axis, the stick will break. The neutral axis is

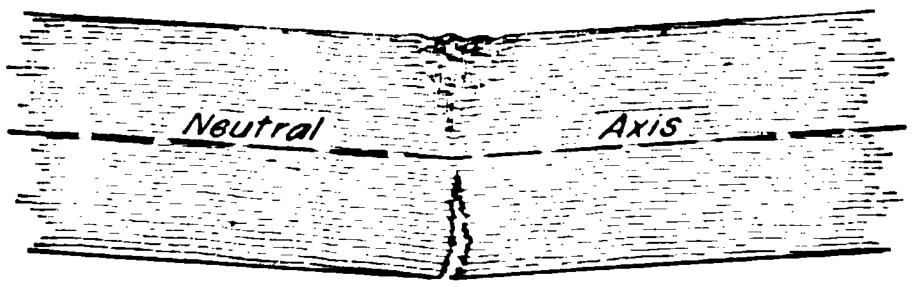


FIG. 5

generally assumed to pass through the center of gravity of the cross-section of the stick.

29. Deflection is the amount that a timber bends when subjected to transverse stresses. For a rectangular beam supported at both ends and loaded at the center, the deflection varies directly as the load.

30. Stiffness.—The resistance that a piece of timber offers to bending is called its **stiffness**. If 100 pounds placed at the center of a stick supported at both ends will bend the stick $\frac{1}{8}$ inch, 200 pounds will bend the stick $\frac{1}{4}$ inch; and 300 pounds, $\frac{3}{8}$ inch, the deflection varying directly as the load. Soon, however, a point is reached where an additional 100 pounds will bend the stick more than $\frac{1}{8}$ inch; this means that the limit of elasticity has been passed. The stiffness of timber does not depend so much on the elasticity of the material as on the dimensions; that is, the ratio of width to depth.

31. Stiffness and Width.—For a rectangular timber supported at both ends and loaded at the center, the stiffness varies as the width. If a piece of wood 2 in. × 2 in. and 4 feet long is supported at both ends, placed with the tangential face up, and then loaded in the middle with 100 pounds, it may be found to bend, say, $\frac{1}{8}$ inch, while a similar stick 2 in. × 4 in. will bend but $\frac{1}{16}$ inch; hence, doubling the width doubles the stiffness of timber.

32. Stiffness and Depth.—For a timber with rectangular cross-section supported at both ends and loaded at the center, the stiffness varies as the cube of the depth. If a 2″ × 4″ stick 4 feet long and supported at both ends is set on edge so that it is 2 inches in width and 4 inches in depth, a load of 100 pounds placed at the middle will bend it

$$\frac{1}{8} \div \left(\frac{4}{2}\right)^3 = \frac{1}{8} \div 8 = \frac{1}{64} \text{ inch;}$$

hence, doubling the thickness multiplies the stiffness of timber about eight times.

33. Stiffness and Length.—For a timber with rectangular cross-section supported at both ends and loaded at

the center, the stiffness varies inversely as the cube of the length of the timber between its supports. If instead of a piece of timber 2 inches wide, 2 inches deep, and 4 feet long, a similar stick of the same cross-sectional area and 8 feet long between supports is taken, the stiffness will be $\frac{1}{(\frac{8}{4})^3} = \frac{1}{2^3} = \frac{1}{8}$ as great, and 100 pounds at the center will cause the timber to bend 1 inch instead of $\frac{1}{8}$ inch. From this, it follows that if the stiffness of a timber 8 feet long is to be the same as one 4 feet long of the same kind and quality of wood, the thickness must be doubled.

34. Stiffness and Grain.—Cross-grained, knotty, or checked timbers are not so stiff as sound wood; and sticks with annual rings set vertically to the load will be stiffer than if the rings are placed horizontally. Sticks partially sawed across the grain are not so stiff as those sawed with the grain, and green sticks are only about two-thirds as stiff as when seasoned. In the same tree, the heavier piece of wood is the stiffer and the butt piece is stiffer than the top piece.

35. Ratio of the Stiffness of Beams.—If the stiffness of a beam supported at both ends and loaded at the center is called 1:

Then that of the same beam, with the load uniformly distributed, will be $\frac{8}{5}$.

Firmly fixed at both ends and loaded at the center, 5.

Firmly fixed at both ends and uniformly loaded, 8.

Fixed at one end and loaded at the other, $\frac{1}{16}$.

Fixed at one end and uniformly loaded, $\frac{1}{6}$.

A cylindrical beam bends 1.7 times as much as one whose cross-section is equal to a circumscribing rectangle.

36. Modulus of Elasticity.—By the elasticity of timber is meant the power of the fibers to assume their original position after being compressed or extended. If the fibers do not return to their original position, they have passed the elastic limit and the timber is deformed. The modulus or measure of elasticity is the ratio of the unit stress, ir

pounds per square inch, to the unit deformation expressed as a fractional part of the length or

$$\frac{\text{unit strength, in pounds per square inch}}{\text{deformation per unit of length}}$$

If 3,000 pounds will bend a beam so that the stress per square inch on the extreme fiber directly under the load is 3,000 pounds and the elongation of this fiber in a length of 1 inch is $\frac{1}{1000}$ inch, the modulus of elasticity will be

$$3{,}000 \div \frac{1}{1{,}000} = 3{,}000{,}000 \text{ pounds per square inch}$$

For different species of wood, this factor varies from about 300,000 to 3,000,000 pounds per square inch. It is independent of the size and shape of the piece and of the method of loading.

37. To Calculate Deflection.—It is often desirable to know beforehand how much a given load will bend a stick. To calculate the deflection of any stick under a load, it is necessary to know the modulus of elasticity. From tests on beams of known dimensions, the deflection being carefully measured, the values of the modulus for different woods have been determined by the formula

$$E = \frac{5\,W\,l^3}{32\,D\,b\,h^3} \qquad (1)$$

for uniformly loaded beams supported at both ends, and

$$E = \frac{W\,l^3}{4\,D\,b\,h^3} \qquad (2)$$

for beams supported at the ends with load concentrated in the middle.

In these formulas, let

E = modulus of elasticity;
W = total load on beam, in pounds;
l = length, in inches, between supports;
D = deflection, in inches, due to load W;
b = breadth, in inches;
h = depth of stick, in inches.

If it is required to find the deflection of a given beam under a given load, formulas **1** and **2** may be transposed so that the deflection will form the left-hand member. Thus,

$$D = \frac{5\,W l^3}{32\,E\,b\,h^3} \qquad (3)$$

for uniformly loaded beams supported at both ends, and

$$D = \frac{W l^3}{4\,E\,b\,h^3} \qquad (4)$$

for beams supported at both ends with the load concentrated in the middle. If it is desired to find the load that will cause a given deflection, the formulas may be transposed so that the load will form the left-hand member.

TABLE II

CONSTANTS FOR DEFLECTION

Species of Woods	Modulus of Elasticity $E = \frac{W l^3}{4 D b h^3}$ per Square Inch Pounds	Approximate Weight That Deflects a Piece 1 Inch	
		1 in. × 1 in. and 12 Inches Long Pounds	2 in. × 2 in. and 10 Feet Long Pounds
Live oak, good tamarack, long-leaved, Cuban and short-leaved pine, good Douglas spruce, Western hemlock, yellow and cherry birch, hard maple, beech, locust, and the best of oak and hickory	1,680,000	3,900	62
Birch, common oak, hickory, white and black spruce, loblolly and red pine, cypress, best of ash, elm, poplar, and black walnut	1,400,000	3,200	51
Maples, cherry, ash, elm, sycamore, sweet gum, butternut, poplar, basswood, white, sugar, and bull pine, cedars, scrub pine, hemlock, and fir	1,100,000	2,500	40
Box elder, horse chestnut, a number of Western soft pines, inferior grades of hardwood	1,100,000	2,500	40

38. Constants for Deflection.—Table II is based on data taken from the Tenth United States Census Reports. The first column gives the species of woods; the second column is derived from the formula for modulus of elasticity; the third gives the weight that will deflect a 1″ × 1″ stick 12 inches long, 1 inch; the fourth gives the weight necessary to bend a 2″ × 2″ stick 10 feet long, 1 inch. The sticks are supported at each end and loaded in the middle. From the third column, it is easy to find how many pounds will cause the same deflection in the same kind of timber of other dimensions. A 2″ × 4″ stick bears eight times as much weight as

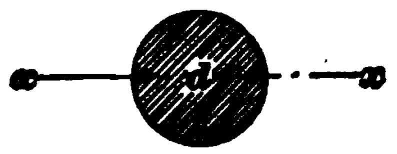
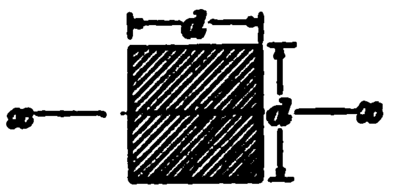
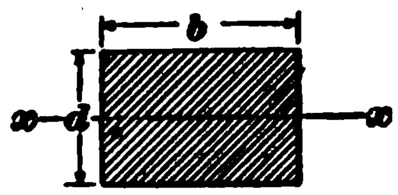
Shape of Cross-Section	Moments of Inertia, I	Section Modulus
Circle 	$\frac{\pi d^4}{64} = .0491\, d^4$	$\frac{\pi d^3}{32} = .0982\, d^3$
Square 	$\frac{d^4}{12}$	$\frac{d^3}{6}$
Rectangle 	$\frac{b d^3}{12}$	$\frac{b d^2}{6}$

FIG. 6

a 2″ × 2″ stick; and a 2″ × 6″ stick twenty-seven times as much, other conditions being similar. A piece of wood 10 feet long is about one-half as stiff as a piece of the same kind of wood 8 feet long, other conditions being similar; a piece of wood 12 feet long is about three-fifths as stiff as one 10 feet long; a piece 14 feet long is about one-third as stiff as one 10 feet long; a piece 16 feet long is about two-ninths, and a piece 20 feet long is about one-eighth as stiff as a 10-foot piece, the same cross-sectional area and the same kind of wood being taken in each case.

39. The **moment of inertia** of a cross-section of any shape is an expression used in determining the resistance

of a stick to bending. The moments of inertia of a circle, square, and rectangle referred in each case to an axis xx passing through the center of gravity of the figure are given in Fig. 6.

In these expressions, d is the diameter of a circle, the side of a square, or that side of a rectangle that is perpendicular to the neutral axis xx of the cross-section, while b is the other side of the rectangle parallel to the neutral axis.

BREAKING

40. Cross-Breaking.—The relative strength of timber depends on how it is supported and loaded. A comparison of the relative strength of beams according to the way they are supported and loaded is as follows:

Beam supported at both ends and loaded with a uniformly distributed load	1
Beam supported at both ends and loaded at the center	½
Beam fixed at one end and loaded with a uniformly distributed load	¼
Beam fixed at one end and loaded at the other	⅛
Beams firmly fixed at both ends and loaded at the center	1
Beams fixed at both ends and loaded with distributed load	1½

When a stick has been bent so as to reach its elastic limit, it requires from 30 to 50 per cent. increase in load before the stick breaks. The load that causes the stick to break represents its transverse strength or its resistance to cross-breaking. *Unlike stiffness, the strength of timber varies as the square of the thickness and inversely as the length and not inversely as the cube of the length.* Thus, if a stick 2 in. × 2 in. and 4 feet long can bear 1,000 pounds, a 2 in. × 4 in. stick of the same kind of wood laid flat and similarly supported and loaded will bear 2,000 pounds before it breaks, and if set edgewise 4,000 pounds. A piece 2 in. × 2 in. and 8 feet long, however, will break with one-half the original load or

500 pounds. The conditions, such as seasoning, and other matters, that influence stiffness also influence resistances to cross-breaking. While conifers excel in stiffness, the better hardwoods develop greater resistance to cross-breaking.

41. Modulus of Rupture.—The measure of cross-breaking refers to the resistance that the fibers offer to cross-breaking; but because of the variability of woods of

TABLE III

TRANSVERSE STRENGTH OF WELL-SEASONED, SELECT PIECES OF WOOD

	Strength of Extreme Fiber $R = \frac{3\,Wl}{2\,b\,d^2}$ per Square Inch Pounds	Approximate Weight That Breaks a Stick	
		1 in. × 1 in. and 12 Inches Long Pounds	2 in. × 2 in. and 10 Feet Long Pounds
Locust, hard maple, hickory, oak, birch, best ash and elm, long-leaved, short-leaved, and Cuban pines, tamarack	13,000	720	570
Soft maple, cherry, ash, elm, walnut, inferior oak, and birch, best poplar, Norway, loblolly, and pitch pines, black and white spruce, hemlock, and good cedar	10,000	550	440
Tulip, basswood, sycamore, butternut, poplars, white and other soft pines, firs, and cedars	6,500	350	280

the same kind, and from the fact that clear small pieces were used to make the tests tabulated, it is customary to use from one-sixth to one-tenth of the figures given in Table III as a factor of safety. The factor of safety means that a beam is to be taken from six to ten times as strong as the calculated loads.

Like elasticity and stiffness, the strength of a stick is expressed in a uniform manner by the so-called **modulus**

of rupture to permit ready estimation of the strength of any piece. This modulus refers to the resistance that the fibers offer just before rupture occurs.

42. Transverse Strength of Seasoned Woods.—The second column of Table III gives the strength of the extreme fiber, which is the same as the modulus of rupture, for the woods grouped in the first column. The figures are for rectangular beams of uniform cross-section supported at both ends, with the loads concentrated in the middle. It is derived from the formula

$$R = \frac{3\,W l}{2\,b\,d^2} \qquad (1)$$

in which R = modulus of rupture, or strength of extreme fiber;
W = breaking load, in pounds;
l = length of beam, in inches, between supports;
b = breadth, in inches, of tested piece of wood;
d = depth, in inches, of tested piece of wood.

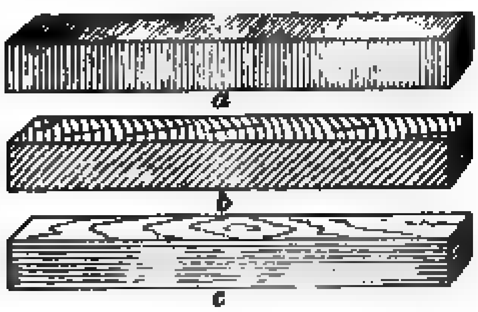
By transposing in formula **1**, formula **2** is obtained:

$$W = \frac{2\,b\,d^2 R}{3\,l} \qquad (2)$$

The third column gives the number of pounds that will break a piece of wood 1 in. × 1 in. and 12 inches long; the fourth column gives the strength of a 2″ × 2″ stick 10 feet long, from which the breaking strength of any given piece can readily be estimated, allowing, however, for defects, which increase with the size of the timber. Thus, if a good piece of pine 2 in. × 2 in. and 10 feet long supported at both ends and loaded in the middle breaks with 400 pounds, a 2″ × 4″ piece set on edge requires 1,600 pounds to break it; a 2″ × 4″ stick 12 feet long will break with 1,300 pounds; one 16 feet long will break with 1,000 pounds, etc.; and if a factor of safety of 10 is allowed, only one-tenth of the foregoing loads is permissible.

43. Compression.—A stick will resist a greater pressure in the direction parallel to its fibers than across them. The perfectly cross-grained stick *a*, Fig. 7, will sustain but

one-tenth the load acting across the length of its fibers that the straight-grained piece *c* will sustain; and it is evident that the piece *b*, which represents the ordinary cross-grained stick, is intermediate in strength between *a* and *c*.



d

FIG. 7

Fig. 7, *d* shows the detrimental influence of knots, since in bending, the lower side of a stick is stretched or in tension, and the upper side compressed. The wood of a knot, like cross-grained wood, offers but little resistance to tension or compression. In the following table is given the compressive strength of common woods in pounds per square inch.

TABLE IV

WEIGHT PER SQUARE INCH CROSS-SECTION TO CRUSH SEASONED WOOD LENGTHWISE

	Pounds
Black locust, yellow and cherry birch, hard maple, best hickory, long-leaved and Cuban pines, and tamarack . .	9,000+
Common hickory, oak, birch, soft maple, walnut, good elm, best ash, short-leaved and loblolly pines, Western hemlock, and Douglas fir	7,000+
Ash, sycamore, beech, inferior oak, Pacific white cedar, canoe cedar, Lawson's cypress, common red cedar, cypress, Norway and superior spruces, and firs	6,000+
Tulip, basswood, butternut, chestnut, good poplar, white, and other common soft pines, hemlock, spruce, and fir .	5,000+
Soft poplar, white cedar, and some Western soft pines and firs	4,000+

44. Shearing.—**Shearing** is a deformation of the wood fibers that may be carried to such an extent as to cause rupture. There are two cases: shearing along the grain

and shearing across the grain. In Fig. 8 (*a*), if a load is placed on *a*, the tenon *b*, by downward pressure, breaks out the piece *c*; in the case of Fig. 8 (*b*), if the shoulder *a* is pushed along *b c*, it is sheared. The resistance of woods to shearing is small compared to their resistance to compression. Wet or green wood shears along the grain about one-third as easily as dry wood. A surface parallel to the rings shears more easily than a surface parallel to the rays. The lighter conifers and hardwoods offer less resistance to shearing than the heavier kinds, but pine shears about one-half as readily as hickory, owing to the construction of the conifer's fibers.

(a)

(b)

Fig. 8

45. Hardness and Shearing.—The properties of hardness and shearing are closely related. Hardwood, as a rule, will offer more resistance to cross-shearing than soft woods. Butt timber, being heavier than top timber, will be harder and will more effectually resist cross-shearing; and seasoned wood will resist more than green wood.

Indentation is one cause of the failure of timbers, and is a common cause of the failure and loosening of tenons.

Very hard woods require 3,200 pounds pressure per square inch to produce an indentation of $\frac{1}{20}$ inch; hard woods, 2,400 pounds; middling hard woods, 1,600 pounds; and soft woods, less than 1,600 pounds pressure per square inch to produce similar indentation.

ROCK PRESSURE

46. Rock-Pressure Theory.—Pressure due to the density of rock increases from the surface downwards until a point is reached where excavations cannot be kept open. In calculating this depth, there are two cases to be considered: In the first, the cavities in the rock are supported by hydrostatic pressure, that is, a body of water opposing the pressure due to the weight of the rock and reaching to the surface. In this case, the specific gravity of the water is to be subtracted from the specific gravity of the rock. The second case is where rock above the zone where no cavity can exist is solid to the surface and has an average specific gravity of 2.7. According to Professor Hoskins, the solution of the problem is reduced to finding the height of a column of rock 1 square inch in area, with a specific gravity of 1.7 in the first case and 2.7 in the second case, reaching from the zone of no cavities to the surface. For the purpose of calculation, the strongest granite having a crushing strength of 24,000 pounds per square inch is taken. Assuming in the first case that the rocks are porous, $1.7 \times .03617 = .0615$ pound per cubic inch as the weight of such rock, .03617 being the weight of 1 cubic inch of water; then .738 pound will be the weight of a column of rock 1 foot high with 1 square inch cross-section. The depth at which cavities would close under such conditions would be $24,000 \div .738 = 32,520$ feet, or over 6 miles.

47. Assuming, in the second case, that the rocks were solid, then $2.7 \times .03617 = .0977$ pound per cubic inch, which gives the weight of a column of rock 12 inches long and 1 square inch base as 1.172 pounds. Hence, the depth at which cavities must close in the second case would be $24,000 \div 1.172 = 20,478$ feet, or nearly 4 miles. Mining will, therefore, be limited by nature to a depth of between

4 and 6 miles according to this reasoning. In mining operations, these principles must be borne in mind, and the size of pillars calculated for their depth below the surface, as these must support the weight and keep open the excavation, a matter very important for shafts. In calculations of this character, because of the uncertainty of the specific gravity of the rock above the excavation and its resistance to crushing, it can be assumed that each foot of rock in depth having a cross-section of 1 square inch will weigh 1 pound and exert that much pressure on the rock below.

48. Weight on Rock Pillars.—Assume that there are several bodies of mineral, each 1,000 ft. × 300 ft., that is, they have an area of 300,000 square feet each. Then at 200, 600, and 1,000 feet depth, the pressures per square inch due to the weight of rock above them, provided that they are horizontal, will be 200, 600, and 1,000 pounds per square inch. If the area of mineral is divided into twenty sections 50 feet wide and 300 feet long, and twenty rooms 15 feet wide and 300 feet long are excavated, the pillars that remain must sustain the same weight as all the mineral before any was mined out. That is, the cross-sectional area of the pillars that equals $35' \times 300' \times 20 = 210{,}000$ square feet has to sustain more than 200, 600, and 1,000 pounds pressure per square inch, according to the depth. From the proportions

$$210{,}000 : 300{,}000 = 200 : x$$
$$210{,}000 : 300{,}000 = 600 : x$$
$$210{,}000 : 300{,}000 = 1{,}000 : x$$

it is found that 286 pounds per square inch is the pressure at 200 feet depth, 857 pounds per square inch is the pressure at 600 feet depth, and 1,429 pounds per square inch is the pressure that comes on the pillars at 1,000 feet depth.

It may be seen from this that, in order to prevent excessive weight coming on pillars, the rooms must be made smaller as depth increases, or larger pillars must be left, which amounts to the same thing. Take, for instance, the last case with the room 15 feet wide and the pillar 35 feet wide and compare it with the room 10 feet wide and the

pillar 40 feet wide. The original area of the mineral would be $50 \times 300 = 15{,}000$ square feet; after the room 10 feet wide was excavated, the area of the pillar would be 12,000 square feet, and the pressure at 1,000 feet depth would be increased from 1,000 to 1,250 pounds per square inch according to the proportion

$$12{,}000 : 15{,}000 = 1{,}000 : 1{,}250$$

In the case of the 15-foot room, the increased pressure on the 35-foot pillar would be,

$$10{,}500 : 15{,}000 = 1{,}000 : x$$

or $\quad x = 1{,}429$ pounds per square inch

This shows an increased weight of 179 pounds per square inch thrown on the pillars by the increased size of room. In mining soft material with a good roof, these pillars must be increased in size rapidly with depth, to prevent the soft material from crushing.

49. Sustaining Excavation Walls.—To prevent the walls of excavations from caving, it is customary either to hold them in place by artificial means or to give the excavations cross-sections, which will assist in opposing collapse. Rock has strength to a degree depending on its character; for instance, sandstone can sustain more weight than slate, and granite more than sandstone, but every rock has its limit beyond which it will rupture.

Rock also has elasticity, and, like timber, will bend before breaking. Assume that Fig. 9 is a cube of rock situated below the surface. The pressures represented by the arrows coming from every direction will balance each other and there will be no movement of the rock particles. The upward, downward, side, and end pressures will be equal; or, if not, the tenacity of the rock is such that no movement takes place. Assume that Fig. 10 is a section of a horizontal

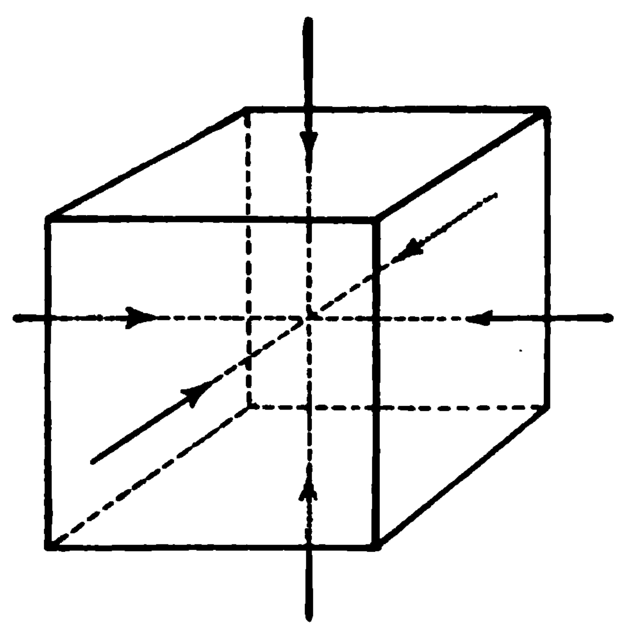


FIG. 9

mine excavation. There will be pressure from each side as shown by the arrows; and if the rock is strong and free from cracks, there will be no need of supports. If, however, the pressure due to the weight of the rock exceeds the strength limit of the rock on any side, that side will require support. Thus, if the pressure on the roof *a* is greater than the tenacity of the rock, the roof should be supported; the same remark would apply to any other side *b*.

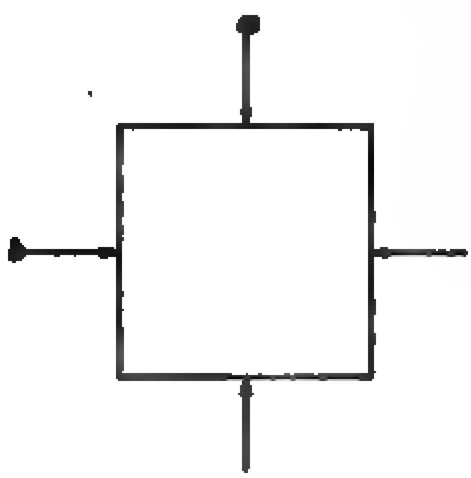


Fig. 10

50. Rock Flowage.—If the side walls are high, compared with the width of the excavation, they may need support; or if the excavation is wide, compared with its height, the floor or roof may need support. A parallelogram is shown in Fig. 11 to represent the section of a horizontal mine excavation. The roof is slate; the bottom, fireclay. The excavation is twice as wide as it is high; therefore, if the pressure is the same from all directions, the top and bottom must sustain twice as much pressure as the sides. It has been noticed in old buildings that stone beams sag; this may also be observed in old cemeteries where marble slabs are supported at each end. This bending is due to what geologists call **rock flowage**; that is, a gradual change in molecular structure. Rocks are not very flexible and if the pressure is such that their tenacity is overcome before rock flowage can occur, the rocks break. Roofs of excavations in stratified deposits must be properly supported or they will bend to a certain extent, as shown in Fig. 11, by the upper dotted line, and then break, particularly if moisture or gas and moisture combined find lodgment back

Fig. 11

of the bend. The dotted lines in Fig. 11 are exaggerated for the sag in the roof, but the line representing the rise in the floor is not, for in some instances the whole excavation is filled with floor rock.

If gas and moisture are not present to break down the roof, it is sometimes broken by the shearing action at the ribs, as illustrated in Fig. 12. The slab *a* of slate or sandstone is bent downwards, while the ribs *b* do not give; this causes a shear along the vertical lines *x* and finally the piece *a* falls.

FIG. 12

The next piece *c* will shear in time and fall, but it will not be so long a piece, as the projections at the ends of *a* form a support for a portion of its length. This continues until the piece *d* forms a step-like arch, where the distance between the ends of the slabs is so small that no bending takes place. Such cases are to be observed in many localities on the surface where water has undermined the rocks. To prevent such occurrences as the foregoing, it is customary, in driving small tunnels in rock or gangways in coal, to give the top less width than the bottom and thus follow nature. A section of a tunnel of this description is shown in Fig. 13, but the arch may be even greater when driven in rock of the same character and not in mineral as illustrated.

FIG. 13

TIMBERS AS SUPPORTS

SINGLE-STICK TIMBERING

51. Timbering Narrow Flat Deposits.—It is customary in mines where the mineral deposits are not more than 8 feet thick, and nearly horizontal, to timber with single sticks termed **props**. The object of standing props is to prevent the roof of bedded deposits from sagging and breaking down eventually, and to hold up loose pieces of rock, or cracked roof. The timbers most generally used as props are round sticks sawed from trees sometimes with bark on and sometimes skinned, preferably the latter; however, sawed or split sticks may be used if they are sufficiently stiff to answer as supports. Props to fulfil this object must be placed at right angles to the floor so that any weight coming from the roof will pass downwards through the center of gravity of the stick, as shown by the dotted line in Fig. 14. A prop is usually cut from 2 to 3 inches shorter than the height of the excavation, and each end is sawed square with the length, in order that the pressure that comes on it may be received evenly over its area at the roof and distributed evenly over its area at the floor. When props are too long, miners sometimes cut them down with an ax, thus leaving an uneven end, which must mash down before the pressure can be properly transmitted through the stick to the floor. This is bad practice, since if the roof sags ever so little it will eventually come down; besides, there is danger of the stick breaking, and while such timbers suggest security they are worse than none for they invite danger.

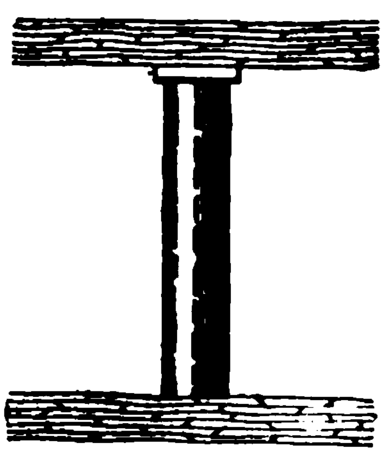


FIG. 14

52. Standing Props.—In mines where the roof is known to sag, props should be placed as soon as the working face is advanced sufficiently to permit, for, to misquote an old adage, "a prop in time will save nine." It is a

common occurrence in some mines where the advantage of promptitude in placing props is not recognized, to see props standing upright supporting only a small weight of rock, the remainder of the weak roof having fallen about them as shown in Fig. 15. In order to connect the prop rigidly with the floor and roof, it is customary to drive a wedge between its top and the roof. These wedges, sometimes termed *caps*, have a flat surface as wide as the top of the post, and of such a length that they will extend about 6 inches beyond the edge of the post. Oak and hard pine sawed from 2 to 3 inches thick are the best woods from which to make caps. It will be noticed, by reference to Fig. 16, that the prop is not stood perpendicular between walls, and from the dotted line that the side *a* is not sustaining weight while the side *b* is sustaining all the weight, although the prop has a cap *c*. Whenever props are stood in this manner, a slight extra weight will bend the prop and either cause it to break or fly out. Whenever a prop is placed in the position shown in Fig. 17, only that portion between the dotted line and the side *b* sustains weight, the remainder of the prop being useless. If the part *b* can support the weight in this position, a smaller prop would answer; the increase in diameter, however, does add a certain amount of stiffness,

Fig. 15

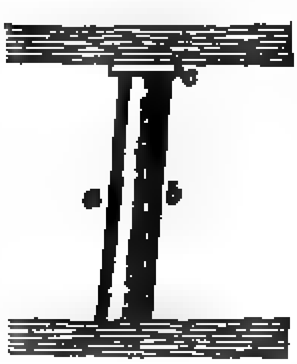


Fig. 16

Fig. 17


Fig. 18

although there might be, under heavy pressure, a shearing off of the part *a*.

Suppose, as in Fig. 18, that there was a bump in the floor that did not permit the lower cross-sectional area of the prop to rest evenly, then the knob would act as a wedge to split the prop through the dotted line, and tamping under the prop would be only partially effective in its prevention. Occasionally when a stick is short, miners will build up a dirt foundation for the prop to rest on and then wedge the prop as tight as possible. This is bad practice, for wedges cannot be driven sufficiently tight to prevent the prop from settling and allowing the roof to sag just a little, which usually amounts to just enough for it to break down. Some miners place a good plank on the floor for the prop to stand on, which is better practice, but even this will only be temporarily secure and not so serviceable as an even floor. In some localities, it is customary to place the butt end of the prop to the roof, as it affords a larger surface for the cap. There is no objection to this if the prop is sufficiently strong and it is placed plumb and firm, but logically the butt of the prop belongs at the floor.

53. Supporting Cracked Roofs.—Props are used to support roofs that are seamy or where there are cracks in evidence, as in Fig. 19. In such situations, the props must

FIG. 19

FIG. 20

be strong and able to sustain the weight of the broken piece of rock above them, for such rocks have no tenacity and will act downwards with their entire weight, or part of their weight,

as shown in Figs. 19 and 20. The miner can sometimes ascertain the existence of cracks by sounding, but if the rock is thick or of a kettle-bottom shape, somewhat similar to that shown in Fig. 19, sounding will not indicate its looseness. This danger cannot be guarded against, but where a break is known to exist it can be propped as shown, unless too large, when the loose rock must be taken down. Where a crack exists, such as that shown in Fig. 20, it is customary to use longer cap pieces than in ordinary prop setting, unless the rock is thick and is known to be jointed and heavy; in the latter instance props should be placed on each side of the crevice. Timbering is varied to meet the conditions of the roof existing at each mine; therefore, in some mines, little will be required; in others, much. But it is always a safe plan to timber under a swag in the roof, as that suggests the bed of a stream of water in the past and an opportunity for the accumulation of water in the future; in fact, in flat-bedded deposits, water is nearly always encountered in depressions of this kind. If water does accumulate above the swag in the roof, it will probably break the roof without warning, owing to the great pressure water exerts, in some cases. While it is not always necessary to calculate the strength of props for mine supports, it sometimes happens that the size of a timber that will support a given weight is required, for which reason the following rules for the strength of wooden props are given.

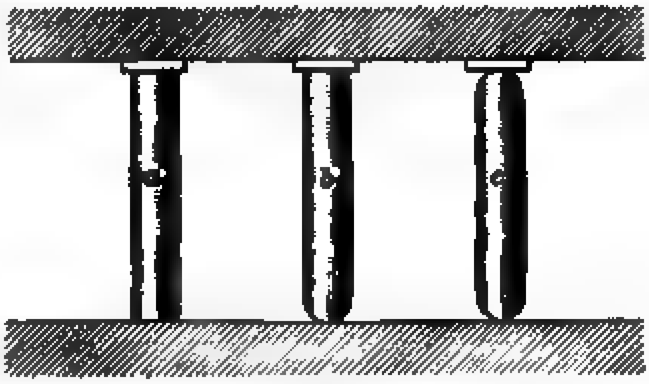


FIG. 21

54. Strength of Props.—It is to be remembered that props are to be set squarely on their ends and to be perpendicular; otherwise, the rules would not apply. It has been proved, mathematically, that the prop *a*, Fig. 21, having flat ends will sustain 4 times as much pressure as the prop *c* having round ends; and $1\frac{7}{9}$ times

as much as prop *b* having one round end; also, it has been calculated that prop *b* is 2¼ times stronger than *c*. The strength of props is increased when their diameters are increased, which is virtually equivalent to shortening a stick up to a certain length when failure occurs simply by compression, and above that length bending assists in its weakening. Stiffness then enters into calculations on the strength of props, for long props of a given diameter will bend more easily than short props of the same diameter. On this account, two formulas are advanced, one for sticks whose lengths are less than 12 times their diameters, and one for sticks whose lengths are more than 12 times their diameters or their least thickness.

The laws that govern the strength of wooden props are but imperfectly understood so that the best formulas are rendered only approximately correct if there are defects or imperfections in the sticks.

For a given diameter, the strength of a prop varies inversely as its length, and its resistance to crushing varies in proportion to the quantity of water it contains; that is, the better a prop is seasoned, the stronger it will become. A long prop may be stiffened by being braced at the center on all sides.

55. Props Shorter Than Twelve Diameters.—In Table V, the safe loads for posts are found in the third column under the heading Compression. The factor of safety given below the headings to the columns is that adopted by railway engineers in bridge construction, although architects use a safety factor of 6 for yellow pine and 10 for other woods. To obtain the crushing strength from this table, it is necessary to multiply the values given by the factor of safety. The safe load for posts shorter than twelve diameters can be calculated from the formula

$$\text{safe load} = \text{cross-section in square inches} \times c,$$

in which c = strength of material as given in Table V.

EXAMPLE 1.—What is the safe working load of a piece of 10″ × 10″ × 10′ seasoned hemlock used as a prop?

SOLUTION.— 10 × 10 = 100 sq. in. cross-sectional area. 120 in. ÷ 10 = 12 times the thickness; hence, the safe load is

$$1,000 \times 100 = 100,000 \text{ lb. Ans.}$$

Tension	Compression	Transverse Rupture	Shearing

EXAMPLE 2.—What is the safe working load of a cylindrical, well-seasoned oak stick 10 inches in diameter and 10 feet long, when the safety factor in the table is used?

SOLUTION.—Cross-sectional area is $.7854 \times 10^2 = 78.54$ sq. in. The safe working load is given in the table as 1,400 lb. per sq. in., the factor of safety being 5; hence,

$$\text{safe load} = 78.54 \times 1{,}400 = 109{,}956 \text{ lb. Ans.}$$

EXAMPLE 3.—What is the crushing strength of a seasoned hemlock post 10 in. × 10 in. and 10 feet long?

SOLUTION.—The cross-sectional area is 100 sq. in., and the length of the post is twelve times its diameter, therefore the safe load is 100,000 lb. The factor of safety used in Table V is 5; hence, the crushing strength is

$$100{,}000 \times 5 = 500{,}000 \text{ lb. Ans.}$$

56. Props Longer Than Twelve Diameters.—When a prop has a length greater than twelve times its diameter, it will bend under a load that a shorter prop having the same diameter would sustain. The Government engineers, in making comparative tests of wood, found that C. Shaler Smith's rule so much used by engineers did not agree with the experiments. The formula deduced by platting the values of all tests made at the Watertown Arsenal is accepted by architects as more nearly agreeing with actual results than any formulas hitherto advanced. For posts having knots and defects or for which the loads are a somewhat unknown quantity—that is, variable—similar to the loads that come on mine timbers, deductions from the results obtained by the use of this formula should probably be made.

Let s = safe load, in pounds;

c = strength, in pounds per square inch, as given in Table V;

l = length of the stick, in inches;

b = diameter of round sticks and least thickness of a rectangular stick, in inches;

$d = \frac{c}{100}$ of wood's strength, as found in the second column of the compression table;

a = cross-sectional area of stick, in square inches.

$$s = \left(c - d \times \frac{l}{b}\right)a$$

Example 1.—What is the safe load, in pounds, for a yellow-pine prop 8 in. × 8 in. and 12 feet long?

Solution.—From the table, c = 1,000; hence,

$$s = \left(1{,}000 - 10 \times \frac{144}{8}\right) \times 64 = 52{,}480 \text{ lb. Ans.}$$

Example 2.—What is the safe load for a white-oak prop 10 inches in diameter and 15 feet long?

Solution.—From the table, c = 900, and the area of the stick is found by squaring the diameter and multiplying the product by .7854; hence,

$$s = \left(900 - 9 \times \frac{180}{10}\right) \times 78.54 = 57{,}962 \text{ lb. Ans.}$$

Example 3.—What is the safe working load, in pounds per square inch, for a sound, well-seasoned, yellow-pine prop 10 in. × 8 in. and 15 feet long?

Solution.—In this instance, the cross-sectional area is not needed; hence, a in the formula is neglected.

$$1{,}000 - \frac{1{,}000}{100} \times \frac{180}{8} = 775 \text{ lb. per sq. in. Ans.}$$

Example 4.—What is the safe working load of a white-pine prop 8 in. × 10 in. and 14 feet long?

Solution.—The smallest side is taken as the breadth and from Table V, c is found to be 700; hence,

$$s = \left(700 - 7 \times \frac{168}{8}\right) \times 80 = 44{,}240 \text{ lb. Ans.}$$

Example 5.—What is the safe working load, in pounds per square inch, for an oak prop 10 inches in diameter and 15 feet long? A deduction of 25 per cent. is to be made on account of a variable load in practice and defects in the timber.

Solution.— $900 - 9 \times \frac{180}{10} = 738$ lb. per sq. in.
for sound timber and

$$738 \times .25 = 184.5;\ 738 - 184.5 = 553.5 \text{ lb. per sq. in. Ans.}$$

57. Pulling Props.—There are methods of mining followed, such as the long-wall and the caving systems, where it is necessary to arrange props so that they may be pulled from under their loads. The object of this is to make the roof sag and break in long-wall mining; and in the caving system of mining, to allow the débris above the timbers to settle uniformly. If the timber cannot be removed by the

dog and chain shown in Fig. 22 or by the use of a block and fall, a hole is bored in it with an auger and a cartridge inserted in the hole and exploded. It has been suggested that pointed props be used in such cases as they will broom and permit sagging to occur. This will prove effective in some cases of longwall mining, provided that the props do not broom at the bottom instead of at the top, and thus become more difficult to pull; but if pointed props were used in the caving system of mining, they would make a comparatively safe system dangerous, since no movement of the top should occur until the end of the room is reached.

Fig. 22

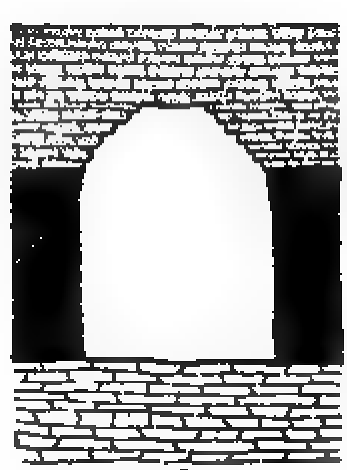
58. Creep.—It is customary in most methods of mining to leave mineral pillars about shafts and about rooms to act as supports for the roof, but it is impossible to formulate a rule to govern the size of pillars that will cover every case and to prevent the floor heaving, or, as it is termed, **creeping.**

The weight on the pillars virtually mashes the strata out from under them into the excavation and thus causes creep. In mines with clay floors, or with floors of such material as will swell when brought in contact with air and moisture, creeping will occur to a very much greater extent than where the material is hard and not easily acted on. Creep, when it once commences, is difficult to stop and may close up the openings, unless the material is removed as it bulges up from the floor. Probably the only method of effectually preventing creep is to leave large pillars at the commencement of operations.

SUPPORTING INCLINED EXCAVATIONS

59. Roof Falls.—Excavations made in inclined deposits require their walls to be supported somewhat differently from those made in flat deposits. By reference to Fig. 23 (*d*), it will be seen that the rock that falls from the roof forms a step-like arch above the excavation; with inclined strata, this dome will have less regularity and will assume the form *a* shown in Fig. 23 (*a*), and the size of the dome will decrease as the inclination of the bed *b* is increased until the perpendicular is reached, when the dome will be zero, as there are no ends to support and gravity acts vertically downwards through the deposit.

(*a*)



(*d*)

Fig. 23

Pillars of mineral left on inclinations must have great tenacity, otherwise they will crumble fast. In some instances, where slippery mineral is on an inclination above 30° from the horizontal, no mining is necessary, the mineral falling away and coming down to the level as fast as it can be loaded and taken away.

60. Roof Pressure.—The pressure on the roof of an excavation is greatest when the bed is horizontal, and least when the deposit is perpendicular, for in the latter case the roof has only its own weight to support, and that may be

lessened by mining. If there is a pressure of 1,000 pounds per square inch on a pillar in a horizontal bed, the pressure would be reduced to .86603 pound per square inch if the deposit were inclined 30° from the horizontal and the pillar were at right angles to the walls, and so on until the deposit was perpendicular, when there would be no pressure from overhanging strata. According to the resolution of forces, the pressure P, Fig. 23 (b), that acts vertically downwards is separated into two components a and b when a deposit c is inclined. The line of greatest stress will not be in the direction a nor yet b, but in the direction of the line d or the resultant of the two stresses due to pressure. In more highly inclined deposits, as represented in Fig. 23 (c), it will be noticed that the stress a perpendicular to the deposit is less than in the former case; while the stress b parallel to the deposit is increased and approaches more nearly the vertical. This will continue until the deposit is vertical when the pressure P will act parallel to the deposit and the pillars will have only their own weight to sustain. The magnetic iron-ore deposits of New York and New Jersey are inclined, and when worked have small pillars of mineral left to support the roof. These excavations are quite free from roof falls, although in some cases they are 2,000 feet deep. The upper sides of these pillars are horizontal, being the floors of former levels, while the under sides are given an angle sloping from the hanging wall to correspond in a measure with the component a, which is at right angles to component b, Fig. 23 (b) and (c).

61. Single-Stick Timbering in Narrow Inclined Deposits.—The principal levels in ore mines are usually timbered for the purpose of support or to guard against accident from small pieces of falling rock. The timbering required in narrow deposits is generally single-stick timbering lagged over. When heavy ground or loose rock is to be supported, cross-timbers, termed **stulls**, are placed over the back or top of levels and covered with lagging. The lagging prevents small rocks from falling on to the level,

and also furnishes a storage place for rubbish. In some cases, stulls are placed as occasional supports for loose rock and are not lagged. In narrow deposits, stulls may be readily handled; but when deposits are wider than 8 feet and the timbers are large, it requires considerable extra work to put them in place. To prepare rests for the ends of stulls, it is customary, whenever the walls will permit, to chisel hitches in the rock. When loose rock is to be supported, it is not customary to cut hitches in the loose piece, but to arrange the stull in such a position that the stress will be compressive along the length of the fibers rather than across the fibers. Some authorities have held that the main rock pressure is transmitted at right angles to the inclination, but this cannot be true where a piece of rock has broken loose from the walls; and in such instances a slight inclination to the rise must be given the stull to prevent the weight of the rock riding to the dip. This feature is illustrated in Fig. 24, where rock pressure on the stull *c* is transmitted to the foot-wall, and where any movement of the hanging wall will tend to make the joints tighter.

FIG. 24

On the other hand, in case the stull was put in at right angles to the walls of the excavation, as shown by the dotted lines *b*, any movement of the hanging wall would throw out the stull. In placing stulls, it is better not to break the hanging wall if it can be avoided, as that will sometimes cause water to flow from the break into the level. It does not matter so much if the foot-wall is broken, since the water will follow the wall, which is on the ditch side.

Where the timbers are used in inclined deposits they have considerable weight to support, and hitches are cut in the

walls in which the ends of the sticks rest. In Fig. 25, the hitches are cut so that when the foot of the stull *c* is inserted in hitch *a*, the other end of the timber will describe an arc to fit into hitch *b*. When the timber rests in the hitches, its ends are tightly wedged.

FIG. 25

62. Single-Stick Timbering in Stopes.—The simplest form of timbering in overhand stoping is shown in Fig. 24, where the stemple *c* forms a support for the lagging. The ore and refuse rock accumulate on this platform, and on this broken material the men stand to work away the vein.

Stulls are also used at intervals in underhand stoping to hold in place loose rock, and to protect the men on the stope below. Stulls, in such positions, are usually lagged over and the lagging covered with refuse, which acts as a cushion, to deaden and transmit the blow from a falling rock over a large surface, thus preventing the lagging from being broken.

Stempels are merely beams supported at each end on the rock walls and as such are subject to bending and breaking loads.

63. Calculating the Strength of Stempels.—The distance between the end supports of a horizontal beam is termed its **span**. If the beam is inclined, the horizontal distance between walls is the span, and not the distance measured along the beam to its supports. If a beam, either horizontal or inclined, is uniformly loaded from one support to the other, it will sustain twice as much weight as if the load was concentrated in the middle of its length; for a uniform load tends to stiffen the beam while the concentrated load tends to bend the beam at its weakest point.

Timbers can be strained very near to the breaking point without serious injury; but, from imperfections not seen, failures sometimes occur at very low limits. **Knots** are the most serious features in beams and practice has demonstrated that beams break at knots, thus exploding the old practical theory that tight knots are not detrimental to timbers.

64. To Find the Breaking Load of a Horizontal Beam Supported at Both Ends.—The safe-load formulas adopted by the United States Department of Agriculture for beams supported at both ends are:

Uniformly loaded beam,

$$S = \frac{4 R b h^2}{3 l} \qquad (1)$$

Load concentrated in the middle,

$$S = \frac{2 R b h^2}{3 l} \qquad (2)$$

in which R = modulus of rupture;
S = safe load, in pounds;
b = breadth of beam, in inches;
h = height of beam, in inches;
l = length of beam, in inches.

The modulus of rupture or extreme fiber stress is given in Table IV, with the factor of safety; and to arrive at the breaking strength, multiply the safe strength in the table by the factor of safety. Railroad engineers use 6 as the safety factor and architects 3.

EXAMPLE 1.—Find the safe working load, in pounds, of a seasoned long-leaved yellow-pine beam 8 inches square and 6 feet long, supported at both ends, using 6 as a factor of safety, the load being uniformly distributed.

SOLUTION.—

$$S = \frac{4 R b h^2}{3 l} = \frac{4 \times 1{,}200 \times 8 \times 8^2}{3 \times 72} = 11{,}378 \text{ lb. Ans.}$$

When a timber is green, the load it will support is about 33 per cent. or one-third less than that it will support when seasoned.

EXAMPLE 2.—Find the safe working load, in pounds, that a seasoned long-leaved yellow-pine beam 8 inches square and 6 feet long supported at both ends will support loaded at the middle, using a factor of safety of 6.

SOLUTION.— $S = \frac{2Rbh^2}{3l} = \frac{2 \times 1{,}200 \times 8 \times 8^2}{3 \times 72} = 5{,}689$ lb. Ans.

Since example 2 is the same as example 1, with the exception of placing the loads, and since a beam uniformly loaded will support twice as much as the one loaded in the middle, the example could have been solved by dividing the answer found for example 1 by 2.

EXAMPLE 3.—(*a*) What will be the safe load for a seasoned white-pine beam supported at both ends and uniformly loaded when its length is 12 feet, its height 12 inches, and its breadth 8 inches? (*b*) What would be the safe load were the stick green?

SOLUTION.—(*a*) Using 6 as a factor of safety,

$$S = \frac{4Rbh^2}{3l} = \frac{4 \times 700 \times 8 \times 12 \times 12}{3 \times 144} = 7{,}467 \text{ lb. Ans.}$$

(*b*) $7{,}467 \div 3 = 2{,}489$ and $7{,}467 - 2{,}489 = 4{,}978$ lb. Ans.

EXAMPLE 4.—Using 6 as a factor of safety, what load would a seasoned oak beam, 12 inches in diameter, support, assuming it to be 12 feet long, uniformly loaded, and supported at both ends?

SOLUTION.—A cylindrical beam is only ten-seventeenths as strong as a square beam whose side is equal to the diameter of the circle; hence, find the load for a corresponding square beam and divide it by 1.7. In this case, $b = 12$; hence,

$$S = \frac{4Rh^3}{3l} = \frac{4 \times 1{,}000 \times 12 \times 12 \times 12}{3 \times 144} = 16{,}000 \text{ lb.}$$

$$16{,}000 \div 1.7 = 9{,}412 \text{ lb. Ans.}$$

SUPPORTING EXCAVATIONS
(PART 2)

MINE TIMBERING

SINGLE-STICK TIMBERING

1. Irregular Walls.—If the walls of an excavation are iard, firm, and fairly regular, the excavation may need scarcely any timbering; but if the walls are irregular or soft, imbers are nearly always required. In Fig. 1 is shown a nethod sometimes adopted when the foot-wall has a different nclination from the hang-ng wall. In this case, the tull has its hanging-wall nd cut on a bevel, to con-orm with the wall and nake a tight joint. The tick is cut 2 or 3 inches horter than the distance etween the walls, in order hat the wedge may be riven under its foot and orm a tight joint. If rranged in regular order, ie stulls are lagged over, nd refuse is placed on the lagging to cushion the blow from alling rock that might otherwise knock out the wedge. . downward pressure will tighten the joints between the tick and the walls.

FIG. 1

2. Measuring the Width of Inclined Deposits.—The width of a vein is measured by a line perpendicular to the foot-wall, and any deviation from this direction will make the width greater. Frequently, veins are measured on a slant, in order to make them seem wider than they are. There is some difference of opinion as to what constitutes a wide or a narrow vein, and, as the matter is arbitrary, a vein whose perpendicular distance from the foot-wall to the hanging wall is more than 8 feet is here considered to be a wide vein. The distance between the walls enclosing a deposit is measured with a slide try rule, and by this means the length of the timber when in position is ascertained. The try rule furnishes a templet with which the length of the stick may be more accurately determined than with a tape measure.

3. Timbering Wide Inclined Deposits.—For supporting the walls of inclined deposits, single sticks are little used if the excavation is made more than 12 feet wide, for the reason that the supports would have to be heavy timbers, which would sometimes be difficult to obtain, besides being more expensive to set up. If, however, single-stick timbering is employed in such places, a block and fall will probably be necessary to raise and swing the heavy stick into place. If there are no timbers above on which to fasten the tackle block, iron pins may be driven into holes drilled into the rock for the purpose. This will require a ladder or temporary scaffolding for the driller to stand on.

In addition to the foregoing, there are certain physical peculiarities in long heavy sticks that have an important bearing on their use. The strength of a timber decreases with its length; that is, a stick that is longer than twelve times its diameter or least depth will by reason of its weight exert an influence on its own stiffness. While it is true that some woods will bend more than others, yet, on account of imperfections, such as checks, knots, and shakes, and the uncertainty of the load that will be placed on the sticks in the mines, the bending factor cannot be definitely determined.

4. Timber Knots.—Whenever timbers are to be raised or lowered with ropes, it is customary to fasten them in rope slings, or secure them with knots. A sling is merely a short piece of rope tied or spliced so as to make a small endless belt. This belt is then looped around the stick, and the hoisting-rope hook attached to the loose end of the loop. When timbers are to be hoisted to a considerable height or lowered down shafts, timber knots are sometimes tied. The knot shown in Fig. 2 (*a*) is a fairly good one, and is much used by bridge builders and framers; it is called the **carpenter's knot.** The knot shown in Fig. 2 (*b*) is also a good one; if properly made, it has some advantage over the carpenter's knot, in that it presents a full hitch for

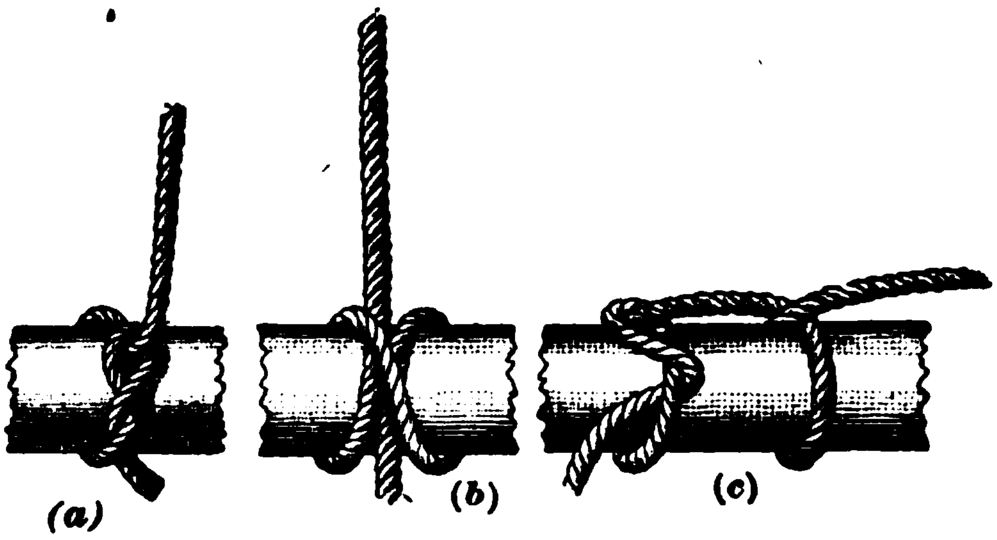


FIG. 2

holding the timber and a loop that offers further frictional resistance to the timber slipping. This knot is much used by riggers and framers, but is not recommended for lowering timbers down shafts unless the rope at the knot is spiked to the timber. Fig. 2 (*c*) is a timber hitch and loop, which offers two surfaces against slipping. Stiff new ropes are not so good for lifting or lowering timbers as pliable old ropes, for which reason slings and knots should be tied with old rope ends, rather than with new ropes.

5. Lowering Timbers Into the Mine.—It is seldom safe to lower timbers into mines with ropes attached to the under side of a cage, even when the loop or knot is spiked to the stick. Long sticks hung in this way turn and swing. Sometimes, the motion is so great as to cause the timber to

hit the shaft walls and catch against some projection. In such cases, the knot may become loosened, and the stick, when freed from the wall, may slip and fall to the bottom of the shaft; or, if the knot does hold, there is danger of the hook becoming detached from the loop and of the stick falling down the shaft. It is best, therefore, in inclined shafts to rest one end of the stick on the car, and in vertical shafts either to rest one end of the stick on the cage bottom or to suspend the timber by a clevis attached to the under side of the cage.

6. Lowering Timbers Down Vertical Shafts.—Where a timber must be taken down a vertical shaft, the best practice is to bore a hole through the timber about 1 foot from the end, then place a clevis, Fig. 3, over the end, and finally insert a bolt through the clevis eyes and the hole in the stick. A nut is screwed on one end of the bolt, and the clevis and stick are suspended from beneath the cage, or from a hook in the hoisting rope if no cage is used. A short piece of rope is spiked to the lower end of the timber, so that the log can be steadied before lowering and can be drawn into the level at the bottom of the shaft.

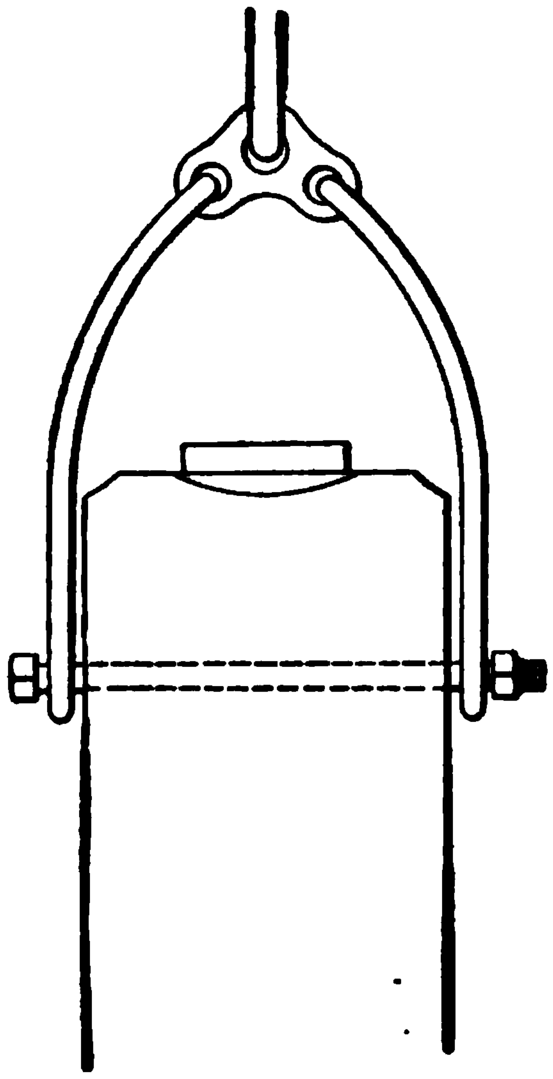


Fig. 3

Sometimes, tongs made with sharp points, like ice tongs, are used in place of a clevis and bolt. The points of these tongs are driven into the sides of the stick to be lowered, while the handle ends are attached to the cage or rope. The weight of the log when suspended causes the tongs to grip the log firmly, but there is danger of the log getting away unless a guide rope is used and extreme care is exercised in raising the log from the ground.

Another system employed is one in which tongs having horizontal teeth to clasp the log and prevent it from slipping

are used. When tongs of this kind are used, a good-sized spike hole should be made in each tong, through which a spike may be driven to prevent the log from slipping if a sudden jerk should take place.

7. Lowering Timbers Down Inclined Shafts. Where skips are used at mines, one method of lowering timbers into the mine is shown in Fig. 4. The top of the

FIG. 4

skip *a b* is removed. Two wheels on an axle, with a clamp to grasp the hoisting rope, are placed on the track above the slope mouth. The timber is run on rollers over the slope mouth and chained at *c* to the axle of the wheels, which, together with the log, are then hoisted up the slope until the loose end of the log drops into the skip. It is necessary to

lower and steady the log into the skip by a rope, as otherwise the skip would be damaged by the log falling from the platform *d*. The arrangement is needed for long heavy timbers only, as several ordinary timbers can be taken down a slope in the skip without the use of the extra wheels.

When a flat car loaded with logs is let down a slope, the logs must be balanced and chained to the car to prevent their slipping. Particular care is needed if the slope is uneven—that is, changes its inclination—for timbers projecting over either end of the car may strike the ground and cause the car tó jump the track.

8. Balanced Timber Skip.—At some mines, special shafts are provided for taking timber into mines, so as not to interfere with the regular hoisting arrangements. A special timber skip with a balance weight is used at some timber shafts and is worked as follows: When the skip is filled with timber at the top of the shaft, the load will descend and raise the balance weight. When the timber skip is unloaded in the mine, the balance weight will descend and raise the empty skip. Thus, according to the load, one unbalances the other. The rope attached to the car and weight passes around a drum provided with a brake, by means of which the speed of hoisting or lowering is regulated. This device does away with the use of an extra hoisting engine for handling timber, rails, and other heavy mine supplies, and is used at mines when the excavations are filled with rock to support the walls.

9. Taking Long Timbers Into Mines.—The problem of finding the greatest length of timber or rail that can be taken from a shaft to a mine level is one that will probably come up at some time or other in connection with every shaft through which long timbers are taken into the mine. To solve this problem exactly, if the length of the timber is to be calculated accurately and the width of the level is considered, is very laborious; it involves the use of higher mathematics, and the results are not always satisfactory. The following formula, while not giving close results in

unusual cases, will be found sufficiently accurate for all ordinary conditions in practice, the error from its use not being more than a few inches.

Let L = length of stick;
w = width of shaft;
h = height of level or gangway;
d = least dimension of stick;

then, $$L = \sqrt{(2w)^2 + (2h)^2} - 2d$$

EXAMPLE.—What is the length of the longest timber 18 inches square that can be taken down a shaft that is 8 feet wide and on to a gangway that is 12 feet high?

SOLUTION.—See Fig. 5. Here, the width of the shaft, 8 ft., is represented by ab, the height of the level, 12 ft., by de, and the least dimension of the stick, 1.5 ft., by jk and lm. It is required to find the longest stick of timber that can be taken around the corner h. Substituting in the letters of the formula,

FIG 5

$$\begin{aligned} L &= \sqrt{(2 \times 8)^2 + (2 \times 12)^2} - 2 \times 1.5 \\ &= \sqrt{832} - 3 \\ &= 28.844 - 3, \text{ or } 25.84 \text{ ft., approximately. Ans.} \end{aligned}$$

TWO-STICK TIMBERING

10. Flat Deposits.—Two timbers so arranged that one will assist the other in supporting the walls of excavations permit of a number of combinations that find application in both flat and inclined deposits. Fig. 6 shows a method of supporting a roadway or preventing a piece of roof from falling into a room. It is termed **cap-and-post timbering**, and can be used wherever the roof is strong. The cap a, which is about 4 inches thick, should not extend more than

2 feet on any side from the center of the post *b*, and should be flush with the roof. The post must be firmly placed and at right angles to the cap.

FIG. 6

11. A similar system of timbering, used in California drift-gravel mines, is shown in Fig. 7. The post is stood in a perpendicular position, and the cap driven into place by blows from a maul. Gravel beds in which such timbering is practiced must be cemented or free from water. Hard pan may be timbered in this way, provided the openings are not wide and there is no water running through the ground. Frozen gravel in the Alaska placer mines does not require timbers of any kind to support it.

FIG. 7

12. In flat deposits, when it is desired to keep a roadway into a room open and safe, the timbering shown in Fig. 8 will answer, provided the mineral is not too soft and crumbly. The collar *b* rests on the post and in a hitch *a* cut into the mineral. Such timbering is intended to stand only while the room is being worked. This method

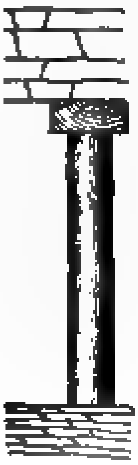


FIG. 8

of timbering is also used in the slicing system of mining, the timber being removed after the gallery had reached its limit, in order that another slice of mineral may be cut from the rib. The object of removing the timbers is to permit the roof to cave, and make mining easier, and also to avoid the danger to which miners would be subjected if the weight should come on the ore and timbers left standing.

13. Timbering Levels.—In ore mines where the foot-wall is weak, the system of timbering shown in Fig. 9 may

(b)

(a)

FIG. 9

be practiced. In this case, the vein is highly inclined, and waste rock is thrown on the lagging to get it out of the way and to act as a cushion in case there should be a rock fall. The collar *a* and the leg *b* are both hitched into the rock and are both lagged, the former to keep stuff off the level, and the latter to prevent the foot-wall from caving. Since there will be more or less pressure from the foot-wall, it will be necessary to brace the leg with the collar *a*; and as there will be considerable weight on the collar from the gob piled on the lagging *c*, it will be necessary to notch the timbers as shown in Fig. 9 (*b*). The leg also must be pressed firmly

against the collar by the wedge *d*. Beams are stiffened considerably by being fastened firmly at the ends, for which reason the end *e* should be fastened as tightly as possible by the wedge *c*. In all cases of mine timbering in which two or more sticks are used, the ends must be wedged tightly and the joints made flush, for should a slight movement occur there is danger of the entire set collapsing. While the flat side of split lagging is often turned toward the excavation and apparently is better so arranged, yet on consideration it will be seen that the flat side should be placed next to the wall to resist the pressure, as the fibers will be in tension on the round side and in compression on the flat side—positions not suited to their greatest strength.

14. Supporting Hanging Walls.—Fig. 10 shows a method of timbering hanging walls that are too soft to carry stulls, or in which it is undesirable to cut hitches. As the ore or rubbish accumulates on the lagging, the cap piece *a*

FIG. 10

FIG. 11

is pressed more firmly against the hanging wall by the stull *b*. Stulls in such sets should not be more than 4 feet apart and should be as nearly on a line as it is possible to place them.

In Fig. 11 is shown another system for sustaining a weak hanging wall. This system is probably better than the one shown in Fig. 10, for the reason that the top stull acts also as a stempel. The stulls *a* and *b* form struts, which tie at the cap piece *c*; hence, any pressure on the cap is transmitted in two directions to the foot-wall. This permits of smaller stulls being used, and at the same time furnishes the requisite strength and support.

15. In Fig. 12 (*a*) is shown a method of timbering that may be adopted when the level is to be permanent and the

(*a*)

FIG. 12

hanging wall and ore are to be supported indefinitely. The joint, Fig. 12 (*b*), has a beveled notch between the leg and

the cap. The leg is shown standing on a plank, Fig. 12 (*a*). This is not a good plan, unless the downward pressure on the collar is greater than the side pressure. If the side pressure is greater than the downward pressure, the leg must be let into a notch cut in the floor. If the floor is too soft to sustain the pressure that comes on the timbers, three-stick timbering must be used in place of two-stick timbering, the third stick acting as a sill for the leg.

16. To gain height and at the same time keep the ore from falling, the system of two-stick timbering shown in Fig. 13 is sometimes adopted. For the timbering to be of much service, the space between the timbers *a* and *b* and the walls must be entirely filled with lagging. The centers between such timber sets should not be more than 5 feet apart.

FIG. 13

17. Timbering Levels in Wide Veins.—Fig. 14 (*a*) and (*b*) shows two simple methods of two-stick timbering in wide deposits. Fig. 14 (*a*) may represent slope timbering in a thick ore deposit, level timbering in a flat deposit, or level timbering in a thick ore deposit. The collar *a* rests in hitches cut in the mineral, and is propped up by the post *b*, which gives it stiffness. By this arrangement the collar is practically divided into two sticks, each of which is supported at the ends, thus making it possible for the collar to support great weight. The post is made less than twelve diameters long, and is placed vertically. An advantage sometimes derived from placing posts in this position in long tunnels is that, if brattice boards *c* are nailed to the posts, the tunnel is divided into two compartments, which will answer the

purpose of airways. Brattice cloth is seldom used in ore mining, but there are many instances where it could be adopted with advantage and economy. Statistics show that metal miners do not live so long as coal miners, which is attributed to the fact that the latter are provided with better air.

In Fig. 14 (*b*), the ore deposit is supposed to be more than 8 feet wide and inclined. The stull *a* rests in hitches cut in the hanging and foot-walls, and, on account of its length, the leg *b* is placed in the position shown parallel with the foot-wall. The leg may be bratticed with cloth or boards, if the space between the wall and the leg is needed for an air passage.

(*a*)

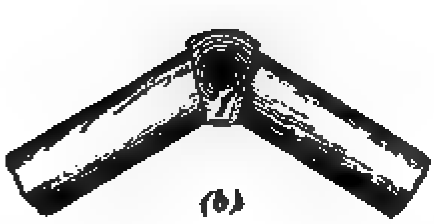
(*b*)

Fig. 14

18. Stull Room Timbering.—An attempt has been made at some iron-ore mines to lessen the amount of timbering by the use of timbers placed as in Fig. 15 (*a*). This is one of the various milling systems of mining that have been practiced; the ore between the dotted lines *b*, *b* is worked into the mill *c*, and drawn off into the cars on the level *d*. The distance between the levels *d* and *e* is about 60 feet, while the width of the room covered

by the stulls is about 20 feet. Timbering of this kind is objectionable in most cases, unless the timbers *a* can be given a pitch of, say, 1 in 3; that is, for every 3 feet along the line *m*, there will be 1 foot rise along the line *n*. This is not enough for heavy ground, since the more nearly the sticks approach the horizontal or the line *m*, the less strength they will have to resist pressure. To explain this, suppose that *n* were an upright post; then, the pressure due to weight would pass through its center. If, however, *n* were inclined, one part of the weight would act perpendicular to the stick, and the other part would be transmitted through the stick to the ground or to the support on which the post was resting. The greater the inclination of the stick from the perpendicular, the less weight would be required to topple it over. The same reasoning applies to each of the sticks *a*; the nearer they approach the horizontal, the greater the pressure from weight must they bear; for, when each is on a high slant, only a small part of the pressure acts perpendicular to the sticks, the greater part being transmitted through the sticks to the heels *o*, *o*. As the inclination becomes less, the perpendicular pressure increases until the sticks are horizontal and become like a beam resting at two ends. There is another objectionable feature to this method where stoping is carried on above such timbering; namely, that it is rarely possible in such cases to distribute the load uniformly on each stick. In case one timber has more load

(a)



(b)

FIG. 15

than the other, it will push the other up at the joint, and eventually both will fall. This may be obviated in some degree by means of a ridge pole *f*, Fig. 15 (*b*), or by a block shaped like a keystone. Ordinarily, such keys make the timbers bind tighter, so that a comparatively light load on the key will balance a heavy load on either stick.

19. Strains on Inclined Beams.—When a stick receives pressure parallel with its length, it is a post or prop and is better able to resist strains than a stick inclined or placed in a horizontal position and loaded at an angle to its length. This is due to the fact that wood is stronger parallel with the length of its fibers than across them. The same principle applies to two-stick timbering, Fig. 15, where the farther the feet *o*, *o* are spread, the less able the sticks are to carry weight. When sticks are in such positions, the pressure is transmitted along the sticks to the walls; then, as the action is equal to the reaction, the pressure is transmitted back again, so to speak, and the two sticks are under compression. If one of two sticks has more inclination than the other, as, for example, *b*, Fig. 13, it receives the greater strain. This is a fact worth remembering, since in mine timbering it is often advisable to use timbers in this manner.

It is an important fact that, however different either of the inclinations or the lengths of two timbers may be, or however different the total strains in the direction of their respective lengths, the horizontal strains caused by the load and the weights of the beams will always be equal on both timbers.

THREE-STICK TIMBERING

20. Slicing and Caving Timber Sets.—Some wide deposits are mined as if they were flat deposits. One system of mining wide beds, known as **slicing and caving,** is shown in Fig. 16. The ore *a* is removed by driving drifts *b*, the excavation being supported by timber sets composed of the legs *c*, *c* and collar *d*. The sets are placed about 6 feet apart from center to center, the drift being 8 feet wide inside

the legs and 7 feet high under the cap. The posts and collar are given a beveled notch. After the drift reaches the walls of the deposit, the track and ties are removed and three round timbers *e* from 6 to 8 inches in diameter are placed

FIG. 16

along the floor of the drift. At right angles to these timbers, saplings, sawmill slabs, or boards *i* are laid. The timber sets are then knocked out and the roof allowed to cave. For the sake of economy, the timber sets are recovered as far as possible, but if they cannot be pulled, holes are bored in them with an auger, dynamite is inserted, and the sticks are broken by a blast. In such mining, the roof must settle, otherwise the work would be unsafe and impracticable. After the ore has been sliced out horizontally to the desired limit, another slice 8 feet high is removed from below. The use of the poles *e* and the lagging *i* now becomes apparent, since they rest on the collar *d*, as shown, and protect the miners.

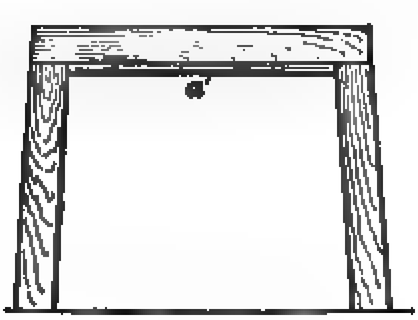


FIG. 17

A better plan than using notch timbers is shown in Fig. 17, where a 2-inch plank *a* is fitted between the posts and nailed

to the under side of the collar. This can readily be knocked off when the timbers are to be withdrawn, and is as strong as the notch and much cheaper to place and pull.

21. Inclined Veins: Strong Walls.—Stulls *a* placed in wide veins with strong walls may be reinforced by the timbers *b* and *c*, Fig. 18, or by struts *b* and *c*, Fig. 19. Such timbering is used in overhand stoping, or above levels. The first method is termed **helped-stull timbering**, the second **saddleback timbering**, the braces practically dividing the long stull *a* into two beams each supported at the ends. When the vein is perpendicular, the stull may be placed horizontally and the legs *b*, *c* carried down to the floor on each side of the level.

Fig. 18

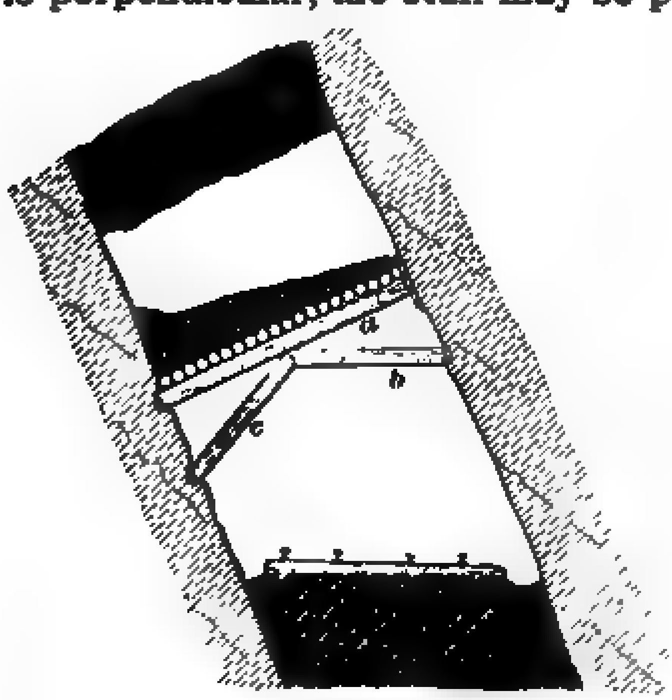


Fig. 19

22. Inclined Veins: One Weak Wall.—In Fig. 20 (*a*) is shown a method of faulty timbering followed in California. The weak wall is a heavy mineralized vein carrying considerable water. The foot-wall *a* consists of compact slate, while the hanging wall *b* is altered diabase. The intention was to notch the collar so that practically the entire pressure would be transmitted to the legs at the joints. Whenever a collar begins to bend at the center, it is forced up at the ends, thus leaving a space

between the collar and the leg. The pressure on the collar is then thrown on the notch at *a*, Fig. 20 (*b*), and will split that piece from the leg, as shown by the shaded portion of the figure. The corresponding piece *b* of the cap, Fig. 20 (*c*), is not much good, owing to the slight resistance it offers, and it, too, soon splits, when, being weakened still further, the stick sags more than ever. The plan of spiking a plank underneath the collar is not safe, since the timbers are round; and, while the beveled notch is much stronger than the squared joint, the method shown in Fig. 21 is probably still better, owing to the fact that it affords a greater bearing surface for round sticks, and because any tendency to kick up at *b* is opposed at *a*, especially when there is side pressure from the wall *b*, as shown in Fig. 20 (*a*). With squared timbers, the beveled joint or the plank nailed to the collar between the legs is better than the joint in Fig. 20.

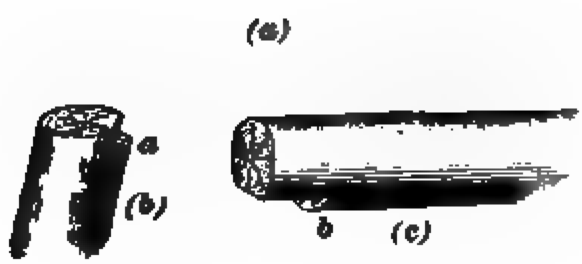


FIG. 20

FIG. 21

23. Inclined Veins: Two Weak Walls.—In Fig. 22 is shown an inclined vein with strong side walls, but of such weak mineral that a leg will sink. The leg *a*, therefore, is made to rest in a notch daped into the sill *b*. The leg in such cases should be inclined as little as possible, not only because it will sustain more weight, but also because the joint *c* at the sill will be stronger. When a post is joined to a sill as shown in Fig. 23, the pressure *a* is divided into the

components *b* and *d* having a resultant *c*. If the leg is given a considerable slope, the tendency is to break off the piece of sill *e*. This is assisted by the heel *h* of the leg acting as a lever, and the weak opposition to shearing offered by timber fibers along their length. Joints between the sills and the legs may be strengthened by the method shown in Fig. 24. The notch is beveled to conform with the inclination of the leg; there is no purchase for the heel *h* to split off the piece *e*. The pressure transmitted through the leg *a* is divisible into two components, one *b* acting parallel to the fibers of the sill, and the other *d* acting across the fibers of the sill. The resultant of these two forces is shown at *c*. If the piece *e* splits off, it must do so

Fig. 22

Fig. 23

Fig. 24

from the point *h*, which would be exceedingly difficult, as the pressure downwards gives the fibers added strength.

24. Limiting Angle of Resistance.—In Fig. 20, the legs of the timber set are inclined so that the pressure

coming on the collar is transmitted equally to both legs. If the legs were placed at different angles, the pressure would come unequally on them, the greater pressure coming on the leg most highly inclined. The friction at the foot of the leg diminishes with the inclination, and if the angle *xzy* is more than 20° the leg has a tendency to slip and increase the horizontal pressure; but the legs will not slip on a level rock surface when that angle, termed the **limiting angle of resistance,** is less than 20°. The friction, which is .4 of the pressure when the limiting angle is 20°, decreases rapidly above this inclination. It is of course possible to block the foot of the leg against the side wall, but even then the horizontal pressure increases rapidly.

25. Sliding Angle of Wood on Wood.—In some cases, legs are tenoned for a mortise in the sill, but this is unnecessary if the angle *o*, Fig. 24, does not exceed 15°, which, according to Morin's experiments, is the limiting angle of contact of oak upon oak when the fibers of the moving surface are perpendicular to the surface of contact and those of the surface are at rest parallel to the direction of the motion.

The friction of two surfaces that have been for a considerable time in contact and at rest is not only different in amount but also in nature from the friction of surfaces in continuous motion. A jar or shock producing an almost imperceptible movement of the surfaces of contact causes the friction of contact at rest to pass to that which accompanies motion. In Fig. 23, if the leg *a* should receive a slight shock, the pressure would be transmitted to the sill in the direction *b* and would probably cause the block *e* to split off.

26. At the New Almaden quicksilver mine in California, the ore body is between a foot-wall of fairly strong serpentized rock and a shattered shaly sandstone hanging wall under which it is dangerous to work. In some parts of the mine, the ore stoped is about 10 feet wide, and the method of timbering is shown in Fig. 25. Heavy stulls *a* are placed in line about 8 feet apart, and immediately above them are placed shorter stulls *b*, which act as legs for the collars *c*

that rest at the other end on a similarly arranged leg. Heavy lagging *d* supports the hanging wall. The lower ends of the lagging rest on the cap of one set, and the upper ends on the lower ends of the lagging of the next set above. For additional security and strength, the two stulls *a* and *b* are bolted together near the top and bottom, as shown by the dotted lines.

FIG. 25

27. Posts and Rough Sets.—In Fig. 26 is shown one method of supporting inclined veins having weak walls with three-stick sets. In such situations, the stulls *a* are termed *caps* and the legs *b* *posts*, although the term *collar and leg* is more appropriate. It is possible with this method of timbering to follow the vein up closely and use the collars as the platform rests. Two-inch planks should be nailed from the posts of one set to the posts of the next set, so as to tie them and prevent their spreading. In some cases, sprags are used for this purpose, but their use involves additional expense for notching and setting, and will probably add no more stiffness than 2-inch planks.

FIG. 26

In stopes of a width requiring only one length of timber to reach across the deposit, as in Fig. 26, and also requiring

posts, it is customary for overhand work to set the sill stulls first, and for underhand work to set the cap stulls first. In all cases, the stulls are cut to such a length as to fit tightly against both walls; or a wedge is driven between one wall and the end of the stull to make it very tight. Each stull is placed in such a position that the upper angle it makes with the hanging wall is greater than a right angle, so that the least setting of the hanging wall will tighten the stull. In cases where the lower stull is set after the upper in underhand stopes, the side posts are set when the stulls have been put in place. In overhead stopes, the posts can be set on the ends of the lower stulls and the upper stull driven down

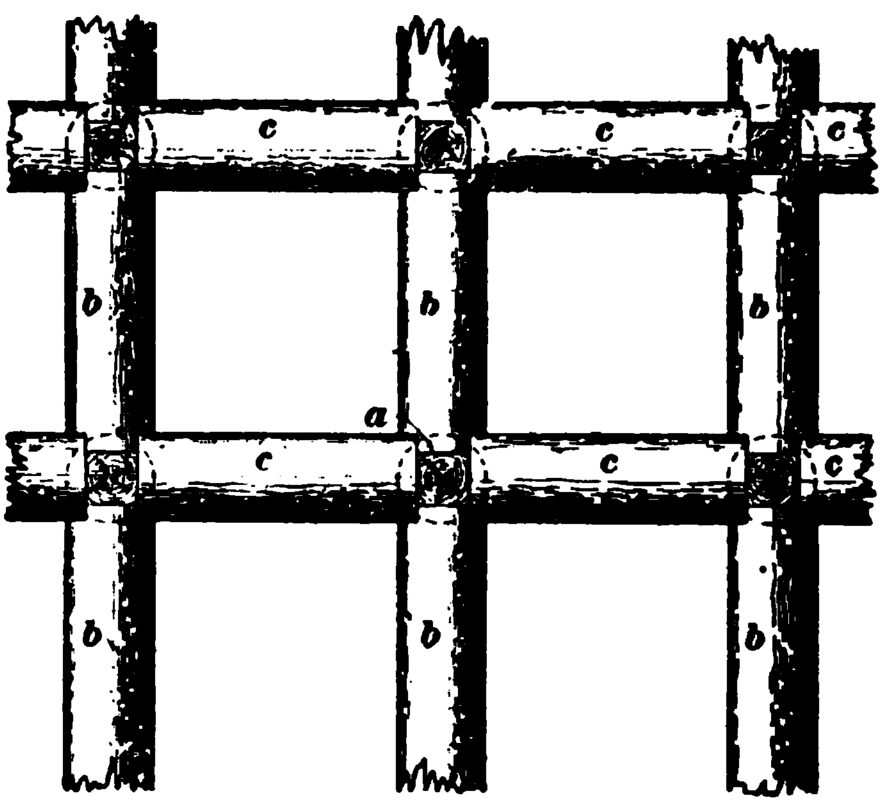


FIG. 27

on the posts. When the side posts have to be set after both top and bottom stulls, it is well to have the top of the posts framed for a gain in the end of the stull, and to drive the bottom of the post into place and secure it with drift bolts. The lower end of the post and the upper side of the stull are not cut for sawed timbers. Fig. 27 is a side view of this timbering when ties *c* are used, *a* being the cap and *b* the post. It will be noticed that the timbers are framed at each end, so that, if lost motion exists at the joints, they will be unsafe for much pressure. It is possible to make the ties smaller than the caps and posts, but by so doing the effective resistance of the latter is reduced.

FOUR-STICK TIMBERING

28. Four-Piece Sets.—A four-piece set consists of two legs, a collar, and a sill for the legs to rest on. Such timbering is required where pressure comes from all sides of the excavation. The legs and collar are given beveled joints if the pressure is about equal from the roof and side; or, if squared timbers are used, a plank may be nailed underneath the collar. In case the pressure is greater from above than from the sides, the joint shown in Fig. 21 or that shown in Fig. 28 may be adopted, the former being preferable for round timbers.

Fig. 28 Fig. 29

In case the pressure is greater from the sides than from above, the joint shown in Fig. 29 is used for squared timbers, and that shown in Fig. 30 for round timbers. The remarks on leg and sill joints in three-stick timbering apply to four-stick timbering except in quicksand and running ground, which require special timber sets.

Fig. 30

29. Timbering Levels.—While the four-piece set can be made of round timbers, it is usually framed from square timbers. When such timbers are to be permanent they are given perfect alinement, and are held longitudinally in place by braces placed between them, or by planks nailed to them. Fig. 31 shows such a set, with planks *a* nailed under the collar to tie the next set. The ditch for levels timbered in this manner is made underneath the sills in the center of the

roadway, and is then planked over. The sills may also serve as cross-ties, to which rails are fastened. In some cases the legs are made long, and compartments are made either above or below the tracks for airways.

30. Drift Sets.—Fig. 32 shows a drift set, which is constructed of round timbers and is quite serviceable. It is divided into two compartments; *a* is an intake airway; *b* is a traveling way; and *c*, the ditch underneath the track, is planked over as shown. The pressure in this case is for

Fig. 31

Fig. 32

the most part from above and from the sides—hence the

necessity of strong lagging. Such sets have about 4-foot centers, and are arranged along the walls as shown in the longitudinal elevation.

31. Connecting Levels.—When stoping by the overhand system, and approaching the level above in which timber sets have been used, it is necessary to make some arrangement to prevent the sets from falling out and letting down the material they are intended to support. The plan shown in Fig. 33 was practiced at the Bimetallic mine, near Phillipsburg, Montana. When the miners are ready to break

(a)

through the floor *a*, stout pieces of timber *b* are placed between the posts *c* and are wedged tightly in place. A heavy piece of timber *d*, long enough to reach across three sets, is placed underneath the caps midway between the posts. The timber acts like a lever having a fulcrum at *e* in the form of a post *f*, the foot of the post resting in the center of the drift on the sill *g*. Wedges are driven in under the collar at *h*, *j*, and *i*, Fig. 33 (*b*), making the joints firm and rigid, after which the rock beneath may be extracted safely, the timber set *k* thus being held in position, and the

superincumbent weight transferred to the points *g* and *h*. As a matter of course, the sill of the set *k* will drop out when the rock on which it rested is removed. The remainder of the timbers may then be connected with those in the set beneath in any feasible manner, thus preventing the loose rock above the caps from falling. The timbers *b* and *d* are then recovered for future use.

32. Taking Up the Timbering of an Upper Stope. When the timbering of any stope is to be taken up on that of another, the task is made very much easier if risers are put through between the levels in advance of the regular stoping. The risers can be timbered with regular square sets, and the stopes may become an enlargement of the risers by the removal of the ore and the addition of other sets. When all the ore has been removed except that immediately under the upper stope, the timbers of one set in the upper stope may be blocked up and supported by long braces reaching to the adjoining sets, after which the block of ground below the set in question is removed and the timbering of the set itself supported from below. When a rise is not carried through and connected with the timbering above, it is practically impossible to guide the posts of the lower stope, and on this account heavy sills have to be used under the timbers that are to be taken up. Where no rise has been put through in advance of the work, it is necessary to break through to the upper stope as carefully as possible.

33. Example of Taking Up the Timber.—Figs. 34 and 35 illustrate one method of taking up square-set timbering in one stope on that of a lower stope. Fig. 35 is a longitudinal section at one side of the car track, while Fig. 34 is a transverse section on the line *A B* of Fig. 35. In the figures, the regular caps are denoted by *c* and the regular legs by *b*; while the regular girths of the square-set timbering in the upper and lower stopes are marked *a*. The set that it has been decided to undermine may be supported by using long timbers *h*, Fig. 35. Near the ends *k* of these

sticks are placed the posts *g*, Fig. 34, which are supported on the regular sills *s*. Wedges are driven between the long timbers *h* and the cap they are to support; this cap in turn supports the legs resting on it. After the timbering has been secured in this manner, the ground between the two stopes is cautiously broken through, and special posts *r* are set on top of the lower timbering, while stringers *d* are placed in such a way as to support the regular sills *s*. It frequently happens, as illustrated, that the posts in the upper set do not come over the posts in the lower set, and in such

FIG. 34

FIG. 35

cases the point in the sill immediately below each post may be supported by the braces *m*, which are heeled to the lower ends of the special posts *r*, as shown. At times, the caps *e*, Fig. 34, on top of the posts *r* are made very heavy, and special stringers are placed across them and under the posts of the upper set, thus doing away with the diagonal braces *m*. After one or two lines of posts in the lower set have come up under the upper set, the work of removing the block of ore immediately under the upper stope progresses with ease, the different timbers being successfully supported by means

of the temporary timbers *h* and their posts *g*. The railroad track and its ties are temporarily supported by means of short posts or braces as they are undermined.

34. Timbering Turnouts.—The method of timbering shown in Fig. 36 may be employed where the width of the vein exceeds 8 feet, and when sticks of sufficiently large diameter cannot be obtained to support the weight, assuming that considerable waste is made in mining. Under the conditions, the stull *a* is reinforced or helped by the strut tie *b* that is held in place by the strut *c*. Such an arrangement will add considerable stiffness and strength to the stull, when the joints between the struts and the key piece are properly made; the stull must, however, be securely wedged at the ends, and made level by wedges driven under its ends, to conform with the strut tie *b*. It will be found that timbering of this description is useful for double-tracked levels or where turnouts are required.

FIG. 36

FIG. 37

SPECIAL TIMBERING

35. Comstock Lode Timbering.—Fig. 37 shows a special system of supporting stulls *a* in a lode about 12 feet wide. The struts *c* and *b* are notched into the posts, the walls being of such a character that they cannot have hitches in them for either the stull or the braces. The sets are placed with 4-foot centers.

On the Comstock lode in Nevada, timbering of all sorts has been tried, including that shown in Fig. 38, which was used to support very heavy ground. The lagging is of heavy planks fitted close together, in order to prevent fine dirt

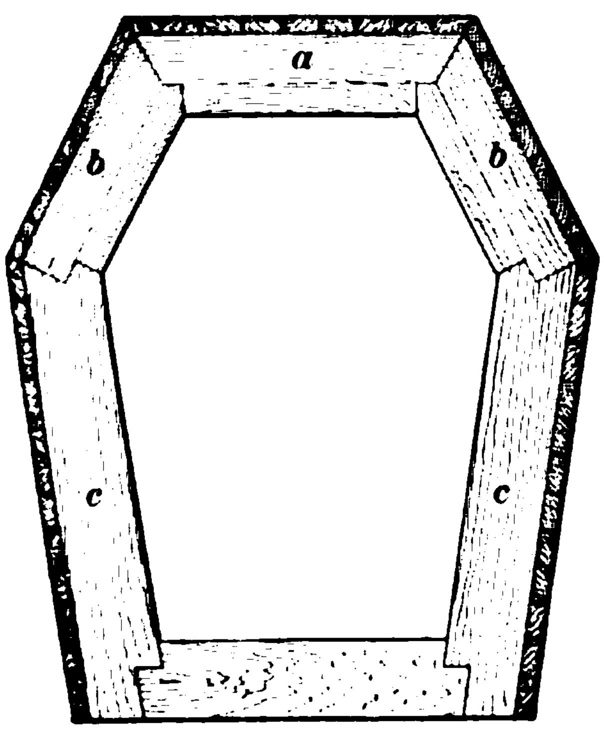


FIG. 38

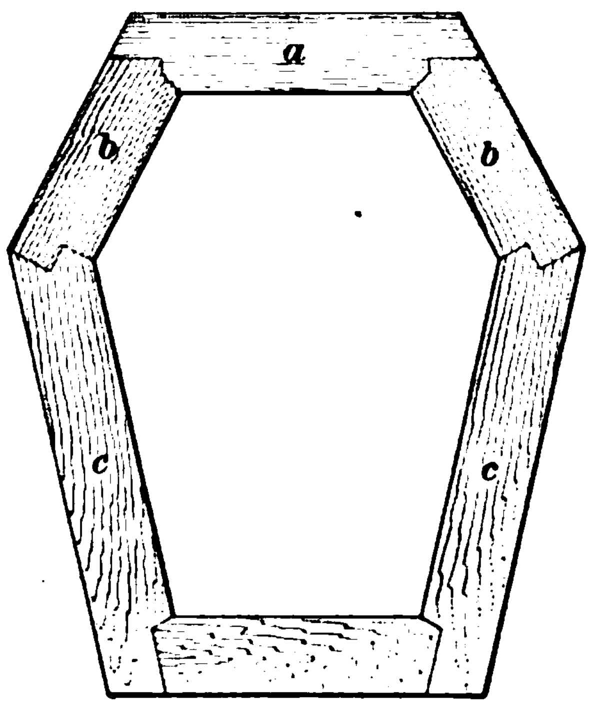


FIG. 39

from running into the excavation. The timber joints are designed to prevent yielding, no matter from which direction the pressure is greatest. The levels are 6 feet high inside the timbers. Probably a stronger system of jointing is shown in Fig. 39, since the full pressure comes on the cap *a*, and is transmitted to pieces *b*, *b*; while in Fig. 38 the pressure on the cap must be sustained mostly by the part *a* above the cap notch, and is then transmitted to the outer part of the leg *b* and so on to *c*, the rest of the sticks acting merely as wedges.

36. There are various methods of timbering wide levels. The method shown in Fig. 40 has many good points in its

favor, since it is strong, easily framed, and quickly placed in position. The post in the center furnishes such strength that this framing can be used for turnouts or double-tracked

FIG. 40

roads, and when partitioned off into two compartments forms intake and return airways for headings. Fig. 41 is a method that has been used where waste rock accumulated, and it was needed to help support the excavation. The legs *a* are placed at an angle, although there is no necessity for the inner leg being so situated, for it would be stronger if placed perpendicular to the sill (see dotted lines), and at the same time the labor of notching would be avoided. The space on the foot-wall side can be used for an airway.

FIG. 41

37. Saddlebacks.—Among the various devices for supporting roofs and stopes in wide deposits is the saddleback

shown in Fig. 42. If the roof is strong and the vein nearly vertical, this system may be sufficient where permanency is not required. It requires less timber than some other methods, but has less stability, and will not be satisfactory

FIG. 42

in heavy ground. Whenever this system is used, the timbers must fit exactly at the joints, since any lost motion will rack the whole frame. If the timbers do not come together flush at the joints, they must be wedged with thin pieces of

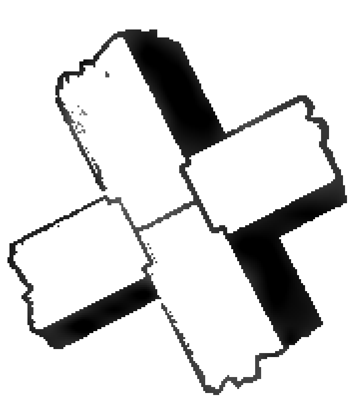


FIG. 43

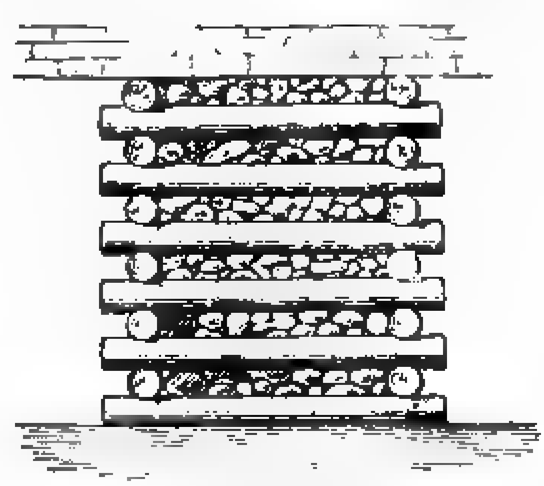


FIG. 44

tough wood. The method of making the joints shown in Fig. 43 is scientifically correct, but the transmission of pressure at an angle without ties to prevent possible movement at the joints is not good practice.

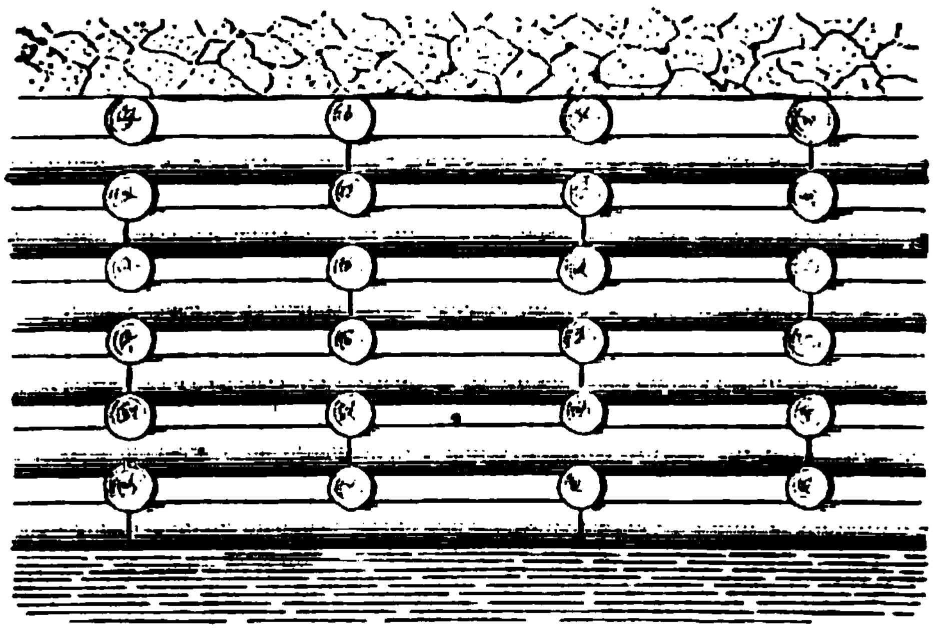


FIG. 45

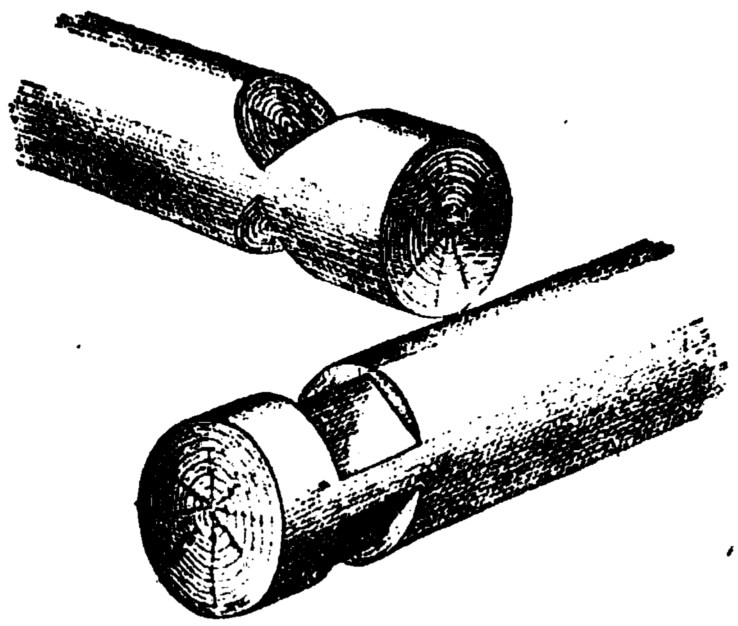


FIG. 46

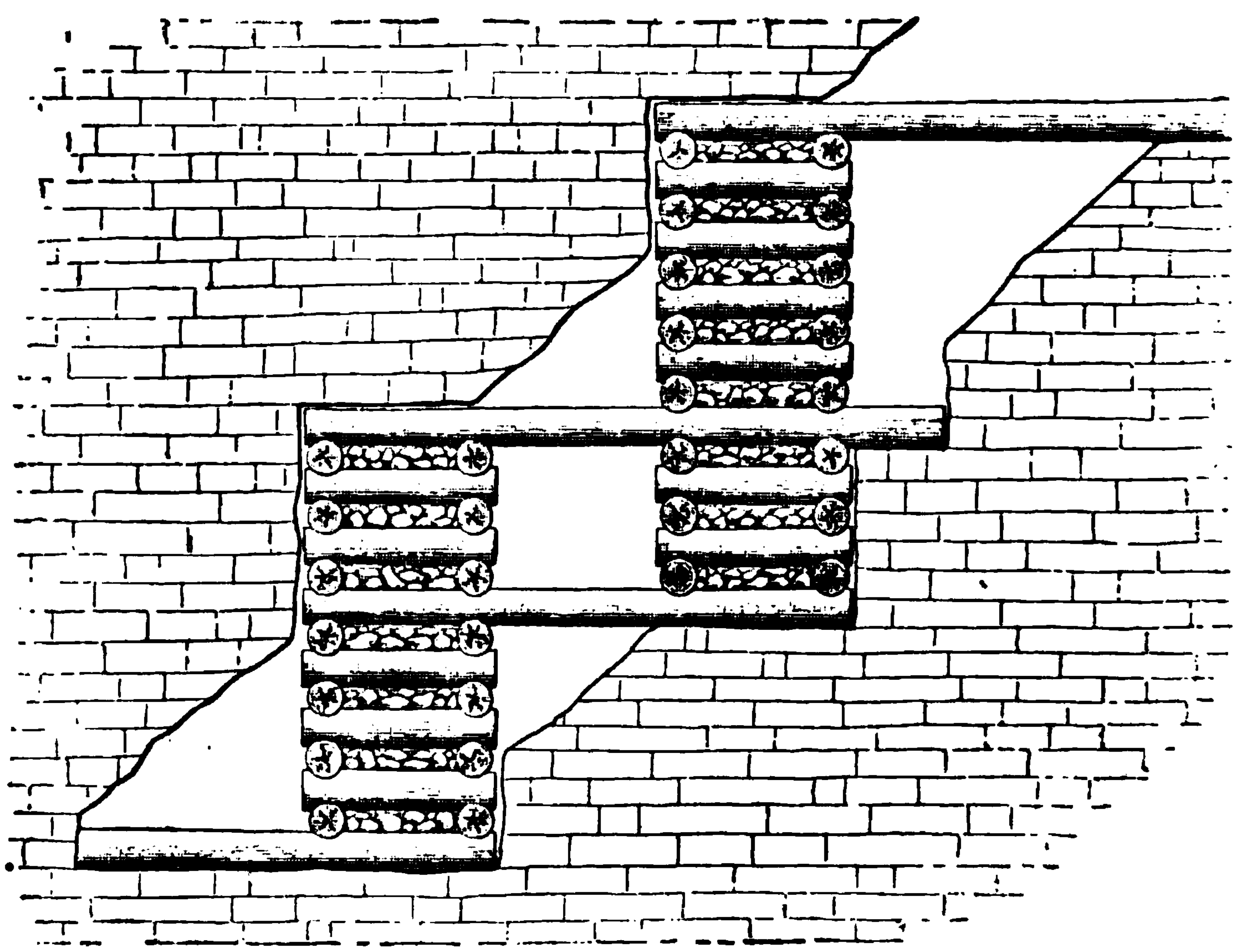


 FIG. 47

38. Cribbing.—When cribbing is constructed and filled in with rock as in Fig. 44, it forms in flat deposits a very useful support for roofs. Where it is desired to keep open a road in a flat deposit, the cribbing may be made continuous, as in Fig. 45. The method of constructing joints for cribbing is shown in Fig. 46. Cribbing is employed in inclined as well as in flat deposits, and is often resorted to for reinforcing other timbering that is being crushed under the weight of superincumbent rock. Fig. 47 shows the method of making crib pens for supporting inclined deposits. The sills for such pens are extended across the deposit, and thus form a tie. It will be noticed that cap pieces to the pens also extend across the deposit and are tied to the adjacent cribbing.

At the Day Dawn mine in Queensland, Australia, a vein of ore 31 feet wide is supported by cribbing.

SQUARE-SET TIMBERING

39. Examples of Square Sets.—The difficulty of supporting the wide and yielding Comstock lode in Nevada led to the introduction of what is known as **square-set timbering.** This method of timbering is now almost universal in wide crumbling veins, and gives general satisfaction, although, like every other good thing, it is frequently misused. In some cases, it is resorted to where it is not needed; in others, it is not properly put in place; and, in still others, it is put in and not properly braced or reinforced. Square-set timbering is not needed in narrow veins; in fact, it is not at all adapted to such ore deposits.

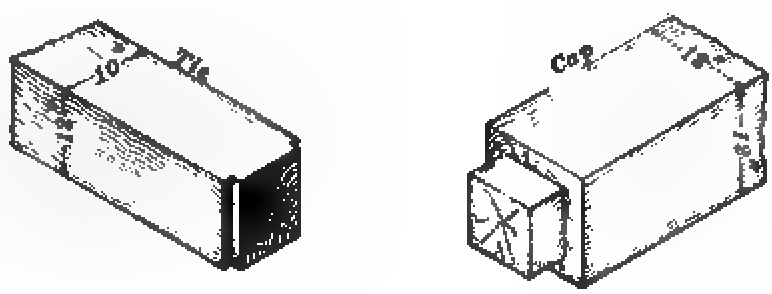
Fig. 48 shows a square set used on the 600-feet level of the Portland mine, in the Cripple Creek mining district, Colorado. The timbers in this case have been squared with a broadax, but frequently they are sawed, if a sawmill is convenient. There is no economy in squaring timbers with a broadax for the small additional strength obtained over sawed or round timbers.

FIG. 48

FIG. 49

Fig. 49 shows square sets used in the Mercur mining district, Utah. The timbers in this case are round. In placing square sets, it is usual to take out a block of ore and immediately replace it with a timber set, in this way obtaining a substantial framework as the ore is worked away and the timber sets are added.

40. Eureka Framing.—The Eureka frame shown in Fig. 50 (*a*) and (*b*) is an example of probably the first square-set framing. It is about as simple as any framing



(*a*)

(*b*)

Fig. 50

intended to resist side pressure, as may be seen by referring to Fig. 50 (*b*), in which it will be noticed that the caps abut. In *A* and *B*, Fig. 50 (*b*), the timbers are all faced 12 inches, while in *C* the ties are faced 10 inches and the caps and posts 12 inches. This arrangement of the ties is for the

purpose of economizing in timber. The length of each timber is about 8 feet, although the timbers are sometimes shortened in the direction of the pressure. Fig. 51 is an illustration of what is known as the **Burlingame frame.** It is somewhat similar to the Eureka frame, but differs in that the posts abut instead of the caps, in order to resist vertical pressure.

Fig. 51

41. Richmond Square Set. The Richmond square set shown in Figs. 52, 53, and 54 is used in Nevada. The framing shown in Fig. 52 is intended to resist both side and vertical pressures. This frame is strong, and offers the greatest resistance to side pressure, its caps abutting. Fig. 53 shows a Richmond set with the timbers coming

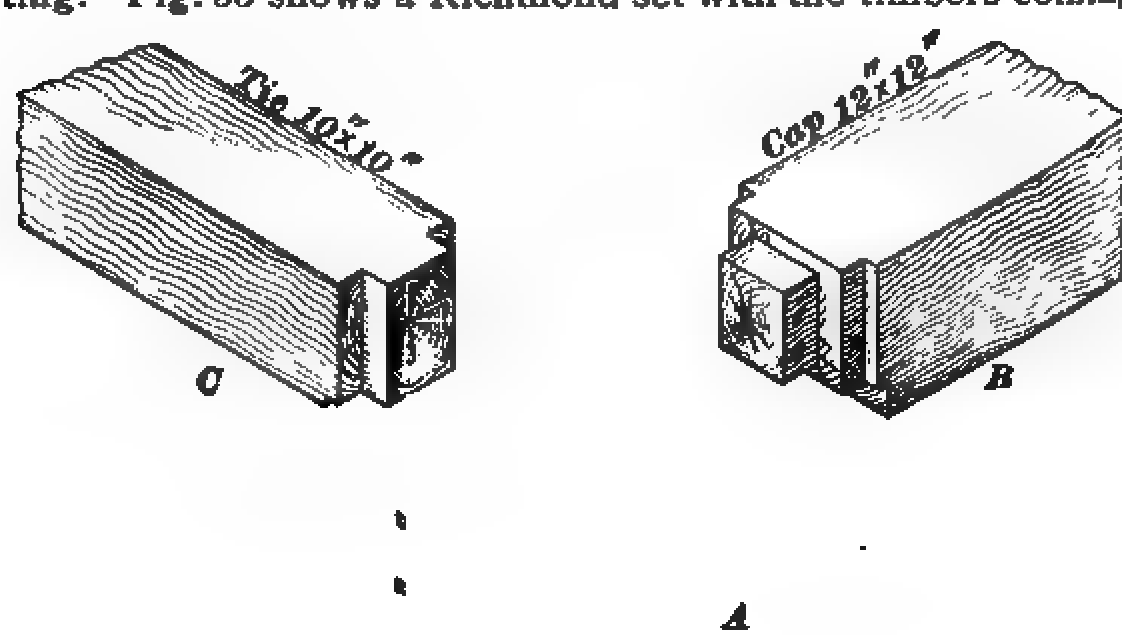


Fig. 52

together from six directions to form a joint, A being the posts; B, the caps; and C, the ties. The latter are $10'' \times 10''$ timbers; the first two are $12'' \times 12''$ timbers. Fig. 54 shows the dimensions and the depth of the notches given the

sticks in this framing. At *A* is shown an elevation of the posts and caps; at *B* is given an elevation of the posts and

Fig. 53

ties with the end of a cap; and at *C* is given a plan of the caps and ties. While this is a good frame, it is not so much

Fig. 54

used as the Eureka, on account of the extra notching and consequent difficulty in making flush joints.

42. Anaconda Sets.—The Anaconda system of square-set timbering was introduced at the Anaconda mines in

Montana. It differs from the Eureka set in having one more notch at each side of the girt, Fig. 55. It opposes pressure from the sides by the caps abutting, but by transposing timbers vertical pressure may be resisted, as in the Burlingame set. The frames are made of 10-inch timbers on lower levels and of 8-inch timbers on upper levels, with reinforcing sets wherever necessary, and at times solid waste filling.

Fig. 55

43. Placing Timbers to Resist Pressures.—Fig. 56 shows square-set timbering for a wide vein that has often been used as an example of how hanging-wall pressure is

Fig. 56

transmitted to the foot-wall. This system was introduced in 1860 at the Ophir mine, Nevada, but has been discarded for square sets without diagonal braces. The pressure in such cases is vertically downwards, and if resolved into two forces

one will act at right angles to the hanging wall and the other, along the hanging wall. The proper system then would be to transmit all vertical pressure by means of frame braces or W-shaped frame braces, rather than by the diagonal braces shown, which have a tendency to knock out the foot of the post and transfer the pressure along the upper side of the caps instead of diagonally as intended. Fig. 57 is an elevation of the same method of bracing, the assumption being that the hanging-wall plates *c* transmit the pressure through the diagonal braces *e* to the foot-wall plates c_1, but that considerable side pressure must still be resisted by the caps *a* and the posts *b*. In the latter case, the braces *c* are out of

FIG. 57

place, since nothing can be more stable than a post set upright with the pressure passing through its axis, and if such posts need assistance they should be reinforced so that the pressure will still be transmitted downwards and not at an angle as shown.

44. Sills for Square Sets.—Instead of using a regular cap piece for the first timber or sill *d*, Fig. 57, on which to rest the posts, the common practice is to frame special sills that will receive two or more posts. Such an arrangement as that shown in Fig. 58 allows of the timbers from a stope below being joined to those above, since it is easier to support the upper timbering, where the sills are continuous

FIG. 58

FIG. 59

and joined under the post as shown at *A*, than it would be when only a cap answered as a sill. These timbers are frequently called **mud-sills**.

45. Nevada Square-Set Timbering.—A sectional elevation of Nevada square-set timbering is given in Fig. 59. The squared-timber posts, caps, and ties, 10 in. × 10 in., stand 4 feet 6 inches apart lengthwise of the vein. The caps are 5 feet long. The posts on the levels are of such a length as to give 7 feet in the clear; while on the stopes they are of such a length as to give 6 feet in the clear. The planks used as staging for the men to stand on when at work are $2\frac{1}{2}$ inches thick, and are moved from place to place as they are needed for advancing the stope. A stope resembles a huge chamber filled with scaffolding from floor to roof. The air circulates between the timbers, thus making the ventilation good, and removing the dust so injurious to the health of miners. Set after set is added as the mineral is worked out, and ladders are put in to go from floor to floor. Chutes are arranged to accommodate the ore and collect it in ore bins at a central point on the level, so that it may be loaded without extra handling. On account of the accessibility to the different faces on a stope, it is possible, when low-grade ore is met with at one face, to mix it at times with richer ore from other faces, and so keep up the grade of the ore to the shipping point. If the timbers show signs of weakening, four or more posts are lined with planks and the bin so formed is filled with waste rock; or, if signs of crushing are noticed, it may be possible to relieve the strain on the timbers by breaking down ore.

46. Reinforcing Sets.—The great value of square-set timbering is due to its strength, to its adaptability to all thicknesses of ore deposits or variations in the hanging and foot-walls, and to the ease with which it may be placed. When the walls are firm, it possesses great strength; but where the hanging wall is weak or softens rapidly on exposure, the system affords only temporary support. When the crush begins, the posts are thrown out of plumb and many

of the timbers splinter. The working or movement of a soft hanging wall indicates the crushing of the timbers in the exhausted portion of the level, and it then becomes a struggle on the part of the miner to remove as much as possible of the ore before the final crush closes out all mining operations on that level. Reinforcing sets are placed wherever there is a sign of weakening, but even these afford only temporary relief. Fig. 60 shows some of the plans that have been adopted for reinforcing weak timber sets. What is termed a *false set* is shown at *a*, and consists of four pieces of timber, although sometimes a cap and two smaller

FIG. 60

posts only are used. Sometimes weakness may be stayed by a diagonal brace as at *b*, particularly where the weakness is displayed at a joint; or, if at two joints, a panel such as *c* may be effective in strengthening those joints. The system of bracing shown at *d* is termed an **N** *set*. The diagonal brace in this set is quite effective in some cases in keeping the set plumb. The **X** brace at *e* is not uncommon where all four joints need reinforcing. The method of placing planks when a panel is to be filled with waste rock is shown at *f*, and the same system will be serviceable for ore bins, particularly for those that are kept well filled, for

in such cases, if the planks bend outwards, they do not by elasticity return to their original position, but will continue to bend until they break.

47. Timbering at Broken Hill Mines.—Fig. 61 is a sectional elevation of square-set timbering at the Broken Hill mines in Victoria, New South Wales, where a wide and

FIG. 61

soft lode is to be stoped away. In the figure, *a* is a temporary platform; *b* is a three-piece false set; *c* is a hand rail; *d* are ladders for reaching the stope; *e* are ore chutes leading to the ore bin *f*; *g* is an ore car; *h* is a loading spout; and the pieces *w* and *i* form diagonal braces.

Fig. 62 is a plan of Fig. 61, showing the platforms *a*, on which the men work; the ore bin *f*, terminating in the loading

apron *h* and the car *g*; and the turntable *t*, which is used in ore mines in place of curves to change the direction of the cars.

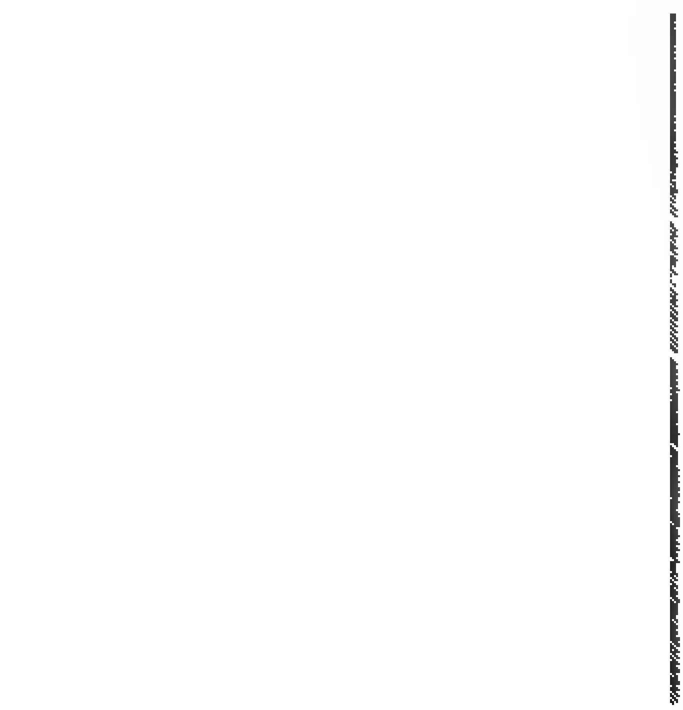


FIG. 62

48. The Loading Chute.—Fig. 61 shows a loading chute timbered in a panel. In Fig. 63 the method of framing these chute sets is shown. The chute floor is to be lined with oak boards or sheet iron, and should have a pitch of at least 32°; in some cases, where the ore is coarse or fine and wet, the floor will often need a greater slope. The apron *a* is usually stationary, but is sometimes hinged so that it may be lowered or raised. The gate *b* may be raised and lowered by a lever. The wheel and gearing shown are somewhat elaborate, but they seem to prove satisfactory for raising and lowering the gate and preventing the rush of ore from the spout.

49. Caledonia Gold Mine.—Fig. 64 is a vertical cross-section of a mine in the Black Hills of South Dakota. The ore bodies *a*, *a* are separated by barren rock consisting of slate *b* and porphyry *c*. The ore was first worked by open cut *e*, followed by an adit level *f*; a shaft was

FIG. 63

Fig. 64

then put down and a hoisting engine put in place. This method of hoisting is practiced quite frequently, and requires special timbering to protect the engine. The 200-foot level was then attacked, and square sets used as the ore was stoped. For some reason, probably for air and exploration purposes, a winze was sunk on the dip to cut the 400-foot level. The ore from this winze was hoisted by a small engine and dumped into an ore bin; this latter arrangement is shown enlarged in Fig. 65. A windlass is usually employed for hoisting automatic dumping skips from winzes. The ore, both from the stope and from the winze, was dumped into the ore bin *m*, and from there trammed in the car *k* to the shaft; then hoisted by the cage to the adit and taken out of the mine. This mine caved in, although the gold-bearing rock *a*, Fig. 64, was hard quartz in chloritic schist. The disaster was due to excavating ore too far in advance of the square sets. The timbers were massive, and were all properly framed, but there is said to have been a disregard of what were considered the minor and unnecessary details in placing them, particularly on the walls and against the roof, where they should have been thoroughly wedged and lagged. As these large stopes were extended, too broad an area was taken out at one time, the weight at length throwing the timbers out of alinement, when the mine caved without a moment's warning, snapping the timbers like reeds and mixing ore, timbers, and machinery in a mass.

It is considered hazardous to remove more than one section of ore four sets wide at one time where such timbering is used (although the breast may extend clear across the vein), since it allows too much weight to come at one time on the timbers in place.

50. Round-Timber Sets.—When square sets were needed in mines producing low-grade ore, it became necessary to reduce the cost of timbering, and round timbers were substituted for squared sticks. Round timbers are stronger in proportion to their area than squared timbers, but they should never be taken into mines with their bark on. To

Fig. 65

further reduce the cost, timber-framing machines and sawmills were introduced to replace hand work at those mines where large quantities of ore were mined and much timber was needed.

Fig. 66 represents square sets formed from round timbers. It shows the worked-out stope of a Michigan iron mine, in which the pressure was not great and the material

Fig. 66

was quickly mined. The ties and caps were of the same size, and were framed alike, as shown in Fig. 67, the dimensions being as follows: *c* and *f*, each 10 inches; *d*, *e*, and *i*, each 2 inches; *a* to suit the size of the posts; and *b* any convenient angle, usually 45°. These sets were so framed that the dimensions, center to center, were 8 feet in all directions, making the posts 7 feet 2 inches long and the other sticks

7 feet 10 inches long. Somewhat shorter timbers with 7 feet centers were adopted in another mine. No diagonal braces were used in the sets, and it was found best to allow the

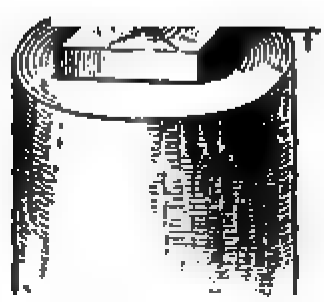


FIG. 67

posts to bite the girths and caps, as by that means the latter timbers were held firmly and had less tendency to spread.

51. Machine-Framed Timbers.—Fig. 68 shows the plan of a machine-framed set of round timbers. The caps *c*, *c* and girths *g*, *g* are notched in such a manner that they have

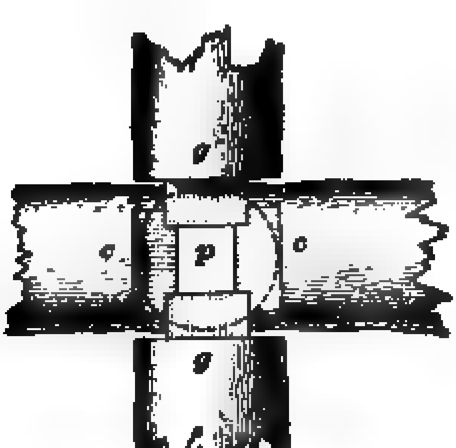


FIG. 68

FIG. 69

bearing joints for the posts whose tenons fit in the space *p*. An elevation of this same framed set is shown in Fig. 69. In this figure, the cap pieces are omitted, in order to show the framing of the girths *g*, *g* and the posts *p*, *p*. The dotted

circle represents the position that the cap would occupy if shown. In Fig. 70 (*a*) and (*b*), it will be noticed that the bottom of the upper post is given a shorter tenon than the top of the lower post. When mine timbers are framed by hand, the joints are always cut a little free to allow for any

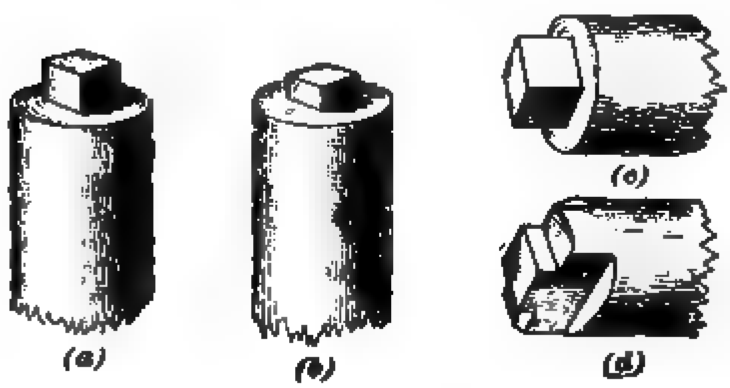


FIG. 70

unevenness in the surfaces; but when the timbers are cut by machinery, the joints are sure to be of the proper size. As timber does not shrink in the direction of its grain, it is evident that, where the posts have tenons that meet, if the caps shrink slightly they will become loose in the space between the shoulders. Fig. 71 illustrates this point; *a* are the posts, and *b* the caps; the posts have tenons *c*, which meet as shown, and the ends framed on the caps just fill the spaces between the shoulders *d*. Now, if the tenons on the ends of the caps shrink, they will not fill the spaces between the shoulders *d*, and there will be open spaces, as indicated by the dotted lines. If the ends of the tenons *e* on the caps *b* fit tightly against the tenons *c* on the ends of the posts, then, when the tenons *c* shrink, there will be open spaces, as indicated by the vertical dotted lines. It is evident that, if the timber is cut green and framed to the exact size, subsequent shrinkage must open some of the joints; but, as most timber is kept damp in the mine, this action rarely causes much trouble, and if its results are feared, the tenons *c* may be made a trifle

FIG. 71

short, so that the shoulders *d* on the posts will continue to bite the tenons *e* on the caps.

52. Timber-Framing Machine.—The faces of the timbers for square sets may be dressed by hand or machinery. Fig. 72 illustrates a machine for framing either square or round timber for square sets. The stick is placed in the machine and secured by means of the dogs *d*. Large saws *a, a* are so placed that they will cut the stick to the exact length over all. The four saws *b* (one of which is not shown) cut down the shoulders on the stick, and the four saws *c* form the vertical faces of the shoulders. After the stick has passed through the saws once, it is drawn back into the position shown, and the entire frame, dogs and all, is

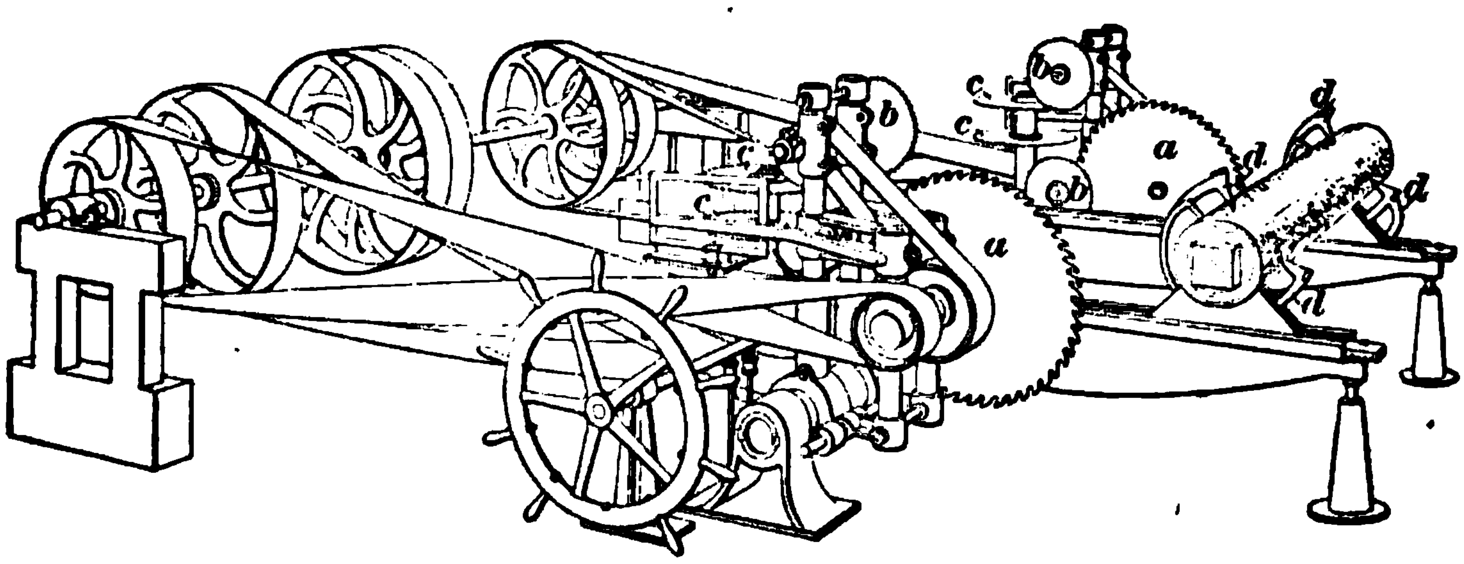


FIG. 72

swung through an angle of 90°, when the stick is fed to the machine once more so as to cut the shoulders on the other side of the tenons. The stick shown in the figure is now finished and ready for removal from the machine. The saws *a* and *b* are cross-cut saws and the saws *c* are rip saws. All the saws can be adjusted within certain limits, so as to cut various styles of tenons. The use of a framing machine makes it certain that the faces of the tenons on both ends of the sticks are perfectly parallel, of the same size, and of the standard length; it is possible, therefore, to frame the joints more exactly to size than with hand framing, and if one end of a timber is larger around than the other the tenons will be of the same size and exactly in the center.

53. Timbering Risers or Winzes.—In risers or winzes, the timbers are placed sometimes skin to skin, and at other times are separated by posts at the corners. Fig. 73 shows a plan of framing square timbers for a rise; the bunton *a* is for the purpose of dividing the rise into two compartments—*b* for the ore and *c* for the manway. These sets are not placed skin to skin, but are separated at the corners by posts tenoned to fit the frames, as shown by the dotted lines. The lining *e* separating the compartments *b* and *c* is made of from 1½- to 2-inch planks. The ladder is supposed to be represented by *d*, but is seldom made as illustrated, the rungs being inserted in holes made in the side pieces of the ladder and not spiked, since in the latter case they are liable to break. Ladders wear fast, and must be inspected frequently.

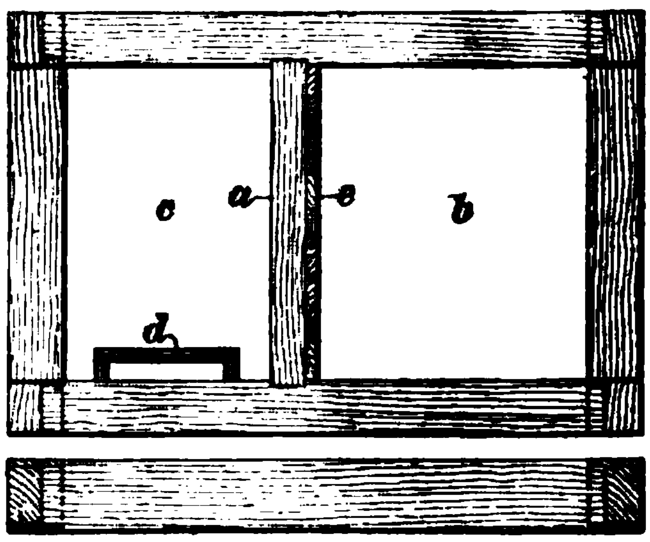


FIG. 73

In Fig. 74 is shown the plan, with a side and end elevation, of a method of timbering that answers admirably in most cases, especially where the winze would need lagging if the timbers were not placed skin to skin. This framing is both cheap and durable, the bunton *a* dividing the rise into two compartments.

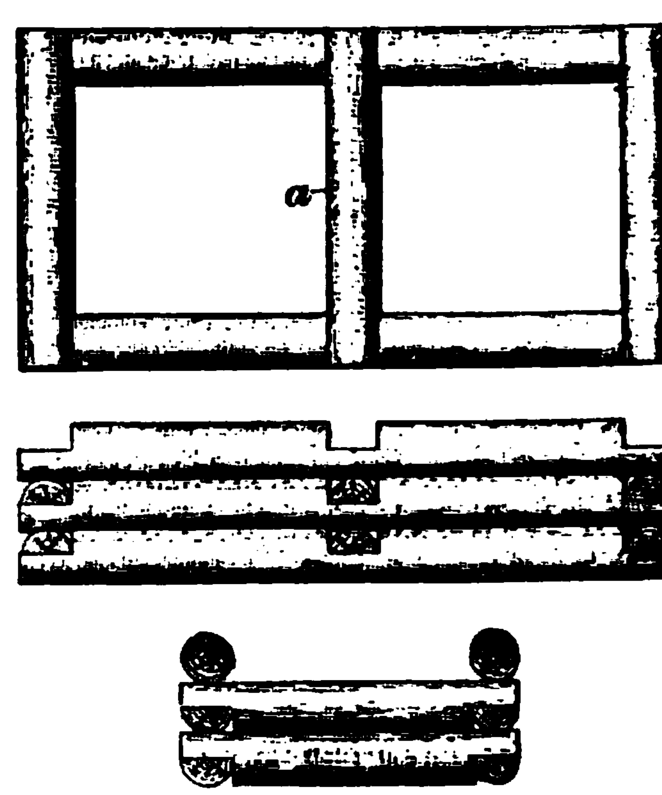


FIG. 74

It is not always necessary to place the timbers close together, and they may be cribbed up as shown in Fig. 75 (*a*), but in such cases it is usual to line the ore chute *b*, Fig. 75 (*b*), with 1-inch boards. The elevation shows the chute *b* to the car, as well as the ladderway *c*. In case the ore becomes jammed in *b*, a man standing on the ladder can start it with a bar. Sometimes the manways *a*

are used as timberways, that is, to raise timbers up to the working levels above. They also form an exit in case anything happens to the level above. Winzes are put down by sinking, and are timbered from above down, the timbers being held in place temporarily by pieces of planks nailed to them.

(b)

TIMBERING SHAFT STATIONS

54. Vertical-Shaft Stations.—At metal mines, it is not unusual to hoist ore from a number of levels, and at these various levels, stations or landings have to be provided. On this account, extensive shaft bottoms are rarely met with in ore mines, although with the use of double-deck cages it may become necessary to provide two landing levels at each station where such cages are loaded.

(a)
Fig 75

Fig. 76 illustrates one style of timbering used in opening out a landing from a shaft that has been timbered with square sets. The ordinary shaft timbering is carried to a point at the bottom of the proposed level, and here the ends of the large stulls shown at *a* are inserted into hitches cut in the sides of the shaft. The shaft and its timbering are then carried far enough to provide a sump, or the sinking may be continued to the next level. When the opening for the

landing is started, it is necessary to remove the shaft lagging, if any has been placed, and after this a 10″ × 10″ post *b* is bolted against the shaft timbering at each side of the proposed opening, and another post is bolted against the regular posts between the hoisting compartments. A 10″ × 10″ cap *c* is then placed across the top of these posts, after which that portion of the wall plate crossing the proposed opening may be sawed out. The station is then timbered

Fig. 76

as shown in the figure, the height of the timbering being gradually brought down to the height of the drift with which it connects. The advantages of this method of opening out are:

1. It is much easier to continue the shaft by means of the ordinary sets than it is to place special sets with long posts where the opening is desired. This is especially true when working in bad ground.

2. If it is desired to push the shaft on to the next level before commencing the opening, this portion of the shaft timbering would be exactly like all the rest; and if later it were found best to change the position of the proposed station, there would be little trouble in introducing the necessary stulls at the desired point and opening out the level in the new position.

55. Inclined-Shaft Stations.—Many ore mines use inclined shafts, and hoist the ore by means of skips. In

FIG. 77

some cases the mining cars are dumped into skips at different levels. Fig. 77 illustrates a plot or station in an inclined shaft where the rock formation is particularly strong and the shaft is small. The shaft timbering is composed simply of stulls *a*, no caps or lagging being employed. The plot is also supported by stulls placed in hitches cut in the rock, and the car track from the level is brought across the top or

hanging-wall side of the shaft, as shown at *b*. The track for the skip is supported on cross-sills *c* that are spiked to the foot of the stulls on the rise side. The shaft illustrated is a two-compartment shaft, the total excavation being 5 ft. × 11 ft., one compartment being used as a hoistway and the other as a ladderway.

In case the walls of the inclined shaft are not hard and strong, the method of timbering a station shown in Fig. 78 may be adopted. In this instance, it is necessary to place the posts *a* at right angles to the hanging wall, and tie them to the caps *b* and the sills *c* and brace them besides with

Fig. 78

the ties *d*. The rails *e* on which the skip runs, sometimes creep, for which reason every second or third sill *c* and cap *b* are made longer than the regular sets in order to rest in hitches in the side walls. To prevent the rails from creeping on steep inclines, it will be necessary to take unusual precautions, such as drilling through the sills *c* into the foot-wall of the slope in order to anchor the sills; or posts similar to *d* may be laid for the rails to rest on and anchored between sills. The inclined shaft should be thoroughly lagged above the caps, and particularly at the station. The station also should be lagged as shown.

Fig. 79

56. Timbering Ore Pocket at Shaft Stations.—In some cases, the ore from the mine cars is dumped into ore pockets provided at the levels. The skips are filled from these pockets and dumped automatically into ore bins at the surface. By this method, tramming is made independent of hoisting for a limited time, since the pockets will contain a sufficient quantity of ore to supply the skip, while the tram cars may be idle; and, on the other hand, if the skip is engaged in serving other levels, the contents of the tram cars can be dumped into the pockets.

Fig. 79 illustrates one method of timbering a station and ore pocket. These pockets are generally arranged in such a manner that the mine cars can come to them from drifts directly in front of the shaft, as at *a*, or from drifts at the sides of the shaft, as at *b*. The timbering of these pocket stations has required a great deal of study, on account of the fact that in many cases they need to be wide. In Fig. 79, it will be noticed that the cap over the drift *b* is composed of three large timbers placed one on top of the other, as it has to carry four of the main timbers supporting the roof of the station. The ore bin is lined inside with planks.

ENGINE-ROOM TIMBERING

57. Underground Engine Rooms.—Although not desirable, it is sometimes necessary in ore mining to use hoisting and pumping engines below ground, for there are so many uncertainties in mining that it sometimes happens that their use cannot be avoided without great expense. When such conditions are met, timbering, steel-beam structures, or masonry are required to make permanent covers, or, as they are termed, engine rooms, to protect the machinery from damage. When timbering is used, the system of framing and the shape of the structure must depend on the width of the room and the height, the latter being greater where steam is used for the machinery on account of the heat radiated.

58. Fig. 80 is a simple strut truss with two rafters *a* and a tie-beam *b c*. The kingrod *d* virtually divides the tie-rod into two beams, and supports the load that comes on the portions *e* and *f* together with its own weight; but it supports also the struts *g* and *h* and part of the loads that come on the rafters *a*. The struts act like posts in affording partial support to the rafters, carrying the strain down to the foot *k* of the kingrod *d*, and from there to the top of the rafters at *i*, and thence down the rafters to the feet *b* and *c*. The action of the struts consists in relieving the rafters of a transverse strain and converting it into a longitudinal or thrust strain in the direction of their length; in other words, the struts add stiffness to the rafters.

FIG. 80

The kingrod has a considerable load to carry, and must be of good bar iron, which with a factor of safety of 3 will sustain 20,000 pounds for each square inch in area. To prevent the bolt from cutting into the fibers of the wood, the rod is provided with a V washer at *i* and a flat washer at *k*. The top of the bolt is headed, and the bottom threaded for a nut in order that the kingrod may perform its intended function and take up sag in the beam *b c*. The truss now is a solid structure that simply rests on whatever foundation may be provided for its ends. If the foundations are posts, then each post must be able to carry one-half the load that comes on the truss. As there are the same dangers from loose rock falling and causing unequal strains, the truss must be lagged. The thrust from the ends of the rafters comes on the tie-beam at *b* and *c*, so that to take up this thrust the rafters must be securely wedged at these points against the rock.

59. The Queen Truss.—Fig. 81 shows another form of truss with two rafters *a*; a tie-beam *b c*; the horizontal straining beam *d*; and two queenrods *e*. The conditions are a roof uniformly loaded along the rafters and the straining beam. The queenrods support all the weight coming on the tie-beam between the parts *f* and *g*, and transfer it to *h i*. The straining beam *d* has equal pressures acting at each end but in opposite directions, producing a strain equal to one of them throughout the entire length of the beam.

FIG. 81

The strains on the rafters *a* will be similar to those on any inclined loaded beam supported at both ends, and the strength of the rafters may be computed by using the formulas given for horizontal beams, taking the horizontal projection of the rafter as its length. The tie-beam *b c* is divided by the queenrods into practically three beams uniformly loaded, besides having a pull or horizontal strain equal to the strain on the beam *d*.

FIG. 82

The truss is practically a solid structure capable of sustaining weight along the roof, and may be braced at the joints by the timber queenposts *j* to hold the straining beam *d* up to its work. The queenrods are made of wrought iron, and are furnished with cast-iron washers. The timbers *a* and *d* must be lagged and the rafters wedged at the points *b* and *c*.

A method of timbering a room in which an engine or a pump is to be placed is shown in Fig. 82. The walls are supposed to be strong where such timbering is employed; if they are not, some such plan as that shown in Fig. 83 may be followed. If, however, the walls are very weak, masonry or a combination of iron and masonry had better be used.

FIG. 83

60. Built-Up Timbers.—It is difficult in many localities to obtain solid timbers of the proper dimensions. When such is the case, smaller timbers of the required depth and number to equal the area of the solid timber needed are placed side by side and bolted together. Timbers built up in this way are not to be considered stronger than solid sticks, although it is possible by this method to select sticks free from imperfections.

61. Ladders.—Shafts having a greater pitch than 14° from the horizontal must be provided with some arrangement so that in case of necessity the men may travel up and down without slipping. From 14° to 40° from the horizontal, stairs with hand rails may answer; above 40° inclination, ladders must be provided. The distance between the top of one step to the top of the next should not be more than 12 inches; while the distance between rungs in a ladder should not be more than 12 inches at the most, and on steep pitches not over 9 inches, particularly if the shaft is deep.

Ladders should not be placed perpendicularly in a shaft, but should be inclined as much as the width of the shaft will permit, and landings should not be more than 20 feet apart in vertical shafts. The sides of a ladder may be made of oak joists or of pine or of spruce saplings 6 inches in diameter sawed through their length in the center. The sides of the ladder should be rounded, in case iron rungs are used instead of oak, particularly in cold climates, since the men must often grasp the sides instead of the rungs. As the work forced on men when they are obliged to climb long, steep ladders is exhausting, it is customary at deep shafts to provide man engines for the men to ride on or else to lower and raise the men in cages or buckets.

TIMBERING IN SOFT GROUND

62. Definition of Soft Ground.—That engineer who in the course of excavating fails to meet with soft material that will not stand without support is fortunate. When the stratum being worked falls into the excavation it is termed *soft ground*. Such material has all grades of weakness, from flaky slate through hard pan to quicksand, which is so mobile that it runs like water. Quicksand being composed of about one-half as much water as sand is the most difficult material to deal with. It exerts a pressure of about .78 pound per square inch for every foot of its height. To deal with such stuff requires experienced timbermen, and is not only expensive but also tedious work. Wet hard pan also causes trouble, because it runs and is heavy; but if such material is permitted to remain in place and the water is drained off slowly, the hard pan can be worked with greater ease than quicksand.

63. Special Methods.—In very soft ground, tunnels have been driven by forcing an iron casing through the clay or other material, the permanent lining being built as fast as the shield advanced.

The pneumatic process also has been applied to tunnel work as a means of keeping soft ground from flowing into

the excavation. This process is very complicated, but was used at Boston in making the subway, and also in tunneling under the North and East Rivers, New York City. It is seldom used in mining, however, and, as the subject is one that belongs to civil engineering, it will not be discussed here.

64. Starting a Drift.—When a drift opening is started, an open cut is made until solid rock is reached, unless the earth is so deep as to make the cut unsafe, in which case the open cut is carried into the bank sufficiently far to gain the desired height of the excavation. As soon as the proper height is obtained, two timber sets are placed as in Fig. 84.

Fig 84

Long lagging poles are driven over the sets into the bank, and the projecting ends are weighted down with rocks, for the twofold purpose of keeping them from sagging at the front end when undermined and of holding the timber sets in place. The sides of the cut are also lagged, commencing at the bottom of the sets and working upwards, and filling in between the lagging and the bank as soon as two or three sticks are in place. The ground is then excavated just enough to admit of another set being placed and lagged, and so on until the solid rock is reached, when a few sets placed under rock cover will probably complete the job. The sets are held in place by pieces of board nailed to two adjacent sets and then thoroughly packed down. Usually,

in such ground, there is a space above the lagging that must be filled before a set can be said to be finished. The hole is sometimes filled by inserting blocks of wood or loose rock above the lagging. Care should be taken, in blocking the sets permanently into place, to see that the load comes on them at the corners and not in the center of the sticks, since in the latter case the sticks would almost certainly break.

65. Timbering Swelling Ground.—Some kinds of rock material, particularly those rocks that are of an argillaceous nature, will swell when exposed to air and moisture. During excavation such ground causes little trouble, but after a time nothing seems able to resist the pressure from its swelling. In timbering an opening through such ground, the best method seems to be to use fairly strong timbers, and to excavate some of the material behind them whenever the swelling begins to exert undue pressure on the sets. The lagging is usually light and open in construction, and by its bending or breaking, the miner is warned that the sets are in danger of being crushed and must be relieved. In some cases, the pressure is not excessive, and very strong timbers set close together will be able to resist it without any special relief. Fig. 85 illustrates a form of timbering and bracing that has been much used in swelling ground, where the strains come on the timbers in an irregular manner. In this figure, A, A are the posts; B, B, the caps; C, the upper sprag, or collar brace; D, the lower sprag, or foot-brace; E, E, the diagonal braces; while F represents the framing of one of the diagonal braces. It will be noticed that

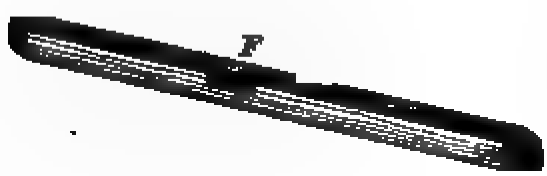


Fig. 85

the braces are notched in the center, so that, when they are in place, they form a scarfed joint. At one mine in Australia where the floor of the levels gives trouble, it is customary to cut holes 18 inches deep on each side of the level. These holes are filled with broken stone, and on top of the stone 3-inch planks 2 feet long are placed for the sills of the timber sets to rest on. This plan has proved successful where other plans failed.

One successful method of holding side swelling ground is to use timber sets of two sizes. The smaller set is made the required width of the level and the wider set is fitted in an excavation between two smaller sets, thus forming shoulders like bay windows on each side of a house. Pointed lagging is driven between the narrow and wide sets in such a way that when swelling occurs the lagging will be pushed into the excavation. Some of the ground is then removed and the lagging again driven into the ground.

66. Bridging, or False Sets.—The term **bridging** is sometimes used to designate a false set of timbers placed outside of the regular set to protect it from the action of the swelling ground. This false set may be composed of as heavy timber as the original set. Such sets were used in the Ontario Mine Tunnel No. 2, near Park City, Utah, and the description given in the following article is substantially the same as that furnished by the superintendent in charge.

67. Ontario Mine Tunnel No. 2.—"Relating to our mode of timbering swelling ground, side pressure is taken care of by the ordinary style of bridging. The timbering over the caps is done with timber of the same size as that used in the set below, but is placed in the shape of an inverted **V**, thus **Λ**, or similar to the rafters of a building, with as many cross-braces as are thought necessary. We used two, also three, side braces—one at the apex, and the other two near the foot. (Corresponding to collar and foot-braces.)

"The foot of this set rests upon the cap directly over the posts. We usually put a thickness of from 1 foot to 2 feet of

blocking on the cap at this point, and then put the **A** set upon the blocks, thus permitting the taking out and raising up of the regular set (by removing a portion of the blocks), without disturbing the upper or **A** set in case your timbers have settled downwards. This can be done at least twice before it becomes necessary to raise the upper set.

"Besides, if we did not have the upper set, the roof would have to be caught up with a false set while raising the regular set; but, having the upper set, it can be easily and quickly caught up with stringers, and the lower set speedily raised without delaying traffic, and without putting up a false set, which takes up much time. The apex of the **A** set is 4 feet above the top of the lower (regular) cap. This gives ample room for men to work above the regular set and ease the roof and take care of their broken material until it can be trammed away. Side bridging is used upon the **A**, the same as upon the regular tunnel sets below. We do not use sills in swelling ground, as we find that it multiplies our troubles. We tried bases 3 feet square under each post, some of rock and some of plank, but the heave of the swell coming very unevenly would tilt posts off their base and wreck the whole set so much that this method was abandoned. The chief pressure on our timbers came from the roof downwards. This was caused by the great weight of water from above, percolating through the brecciated strata; when this ground is relieved by drainage, I am satisfied that the pressure will be less. Regarding soft running ground, the same methods were adopted as are employed elsewhere in such cases, except that, where the ground was very heavy upon the spiling and they could not be driven in by hand, we shod them with $1\frac{1}{2}$-inch iron and drove them with a $3\frac{1}{2}$-inch Ingersoll drilling machine with 90 pounds of air, which proved quite successful."

This tunnel was completed in the latter part of 1894, and has a total length of about 15,000 feet. The total height of the tunnel is $8\frac{1}{2}$ feet, divided as follows: tunnel proper, 6 feet high, 4 feet wide at the top, and 5 feet on the floor; below the floor or car track is a water ditch $2\frac{1}{2}$ feet deep

and $5\frac{1}{2}$ feet wide at the bottom. Although in some parts of the tunnel there was good dry ground, in other parts there was swelling ground and floods of water. The flow amounted to as much as 10,000 gallons of water per minute for some

Fig. 36

time, and on one or two occasions, floods occurred that caused a temporary cessation of work.

The car track was spiked to a 3-inch plank, which rested on $6'' \times 10''$ sills. Mules hauled the mine cars out of this tunnel in trains. At distances of 1,000 feet in the tunnel, sidings were put in to allow trains to pass each other.

Two air drills were kept going in the face on 8-hour shifts. The 20-inch ventilating pipe, Fig. 86, was connected with a Root blower and used sometimes for ordinary ventilation, but its chief function was to clear away smoke after blasting; at such times it was operated as a suction fan and cleared the face in a few minutes.

Fig. 86 is a cross-section giving a front view of the upper and lower sets. *A* are the posts of the regular (lower) set; *B* is the cap of the same; *E* are the rafters or upper set,

Fig. 87

which have a cross-brace *F*; the bridging of the upper and lower sets is represented by *G*, and the lagging by *H*. The sills *J* are to support the 3-inch plank *K*, which forms the floor of the drift and on which the car track is laid. The dotted squares show the position of the horizontal braces or sprags extending from this to the next set. The set is lagged in the usual way, except that the lagging is behind the bridging. The bridging really constitutes a shell, which

partly protects the set proper and greatly facilitates the excavation of ground in the case of serious swelling. If only the usual tunnel sets are employed, there is more danger of their being totally crushed, and in excavating to relieve the pressure the roadway would be blocked, thus interfering with the traffic. *C* and *D* represent blocking placed over the regular set and below the upper set and its bridging.

68. Center-Post Braces.—Fig. 87 is a cross-section of the timbering used in the Sutro tunnel, where there was considerable water to care for, and the top pressure was severe. It was presumed that one post under the collar in the middle of the tunnel would not be sufficiently strong to withstand the top pressure, or rather would not add stiffness enough to prevent the collar from sagging.

69. To Determine the Size of Mine Timbers.—The sizes of timbers are chosen in mine work by mental estimation rather than by arithmetical calculation; nevertheless, experience and science enter into that estimation. While the size of a timber is chosen with a general idea of the strain it is expected to bear, yet it is considered better practice to increase the number of timbers rather than their size. The principal objections to heavy timbers are the great difficulty of handling and placing them underground and their cost. The timbers used for square sets may vary in size from 8 in. × 8 in. to 24 in. × 28 in., the latter size being unusual on account of the difficulty in obtaining them. California, Michigan, and New Jersey mines use heavy round timbers from 20 to 30 inches in diameter on wide stopes; but in places where such large timbers are not available, the tendency is to use timbers 12 in. × 12 in. and even smaller, and then depend on filling to help sustain the pressure. The latter plan is probably the better.

70. Strength of Stulls.—The size and number of stulls required to sustain the weight of waste ore will depend on the width of the vein and the height to which the waste is likely to accumulate. It is better to increase the number

of stulls rather than their size, for the reason that heavy stulls are more expensive to handle and place and they require heavier lagging than lighter stulls placed nearer together. With good walls, stulls 7 feet long, 12 inches in diameter, and 30 inches apart are calculated to sustain 60 feet of waste or broken ore in a vein standing at a high angle; then, as the strength decreases directly as the increasing length, a stick 10 feet long and 12 inches in diameter will sustain but 42 feet of waste, and a similar stick 12 feet long will sustain only 35 feet of waste.

71. Reinforcing Stulls.—After waste has remained in place for some time, it will pack and thus relieve the stulls

FIG. 88

from strain. In some instances, the pack will become so firm that it will retain its solidity after the stulls have fallen out or have been removed. This need not be expected in wide veins; in fact, if the packing starts to move in any vein, it will in all probability continue. When it is necessary to catch up settling stulls or caps, a stout post *a*, Fig. 88, which will just fit under the sinking timber *b*, is placed in position, and four wooden wedges *c* are driven in between the stull or cap and the sinking timber.

MASONRY AND METAL SUPPORTS

MASONRY SUPPORTS

72. Masonry Combination.—Masonry alone, masonry in connection with timbers, or masonry in connection with iron or steel is now being extensively used in Europe, and

FIG. 89

also to some extent in the United States, but more particularly in coal and iron-ore mines where permanent work is desired. Only large mines keep their shaft stations open for a long time; and as the deposits in which many ore

mines are located are liable to squeeze and crush the supports, it follows that if the latter are timbers it is much easier to repair them than when of masonry or steel. Masonry linings for tunnels primarily cost more than timber; but, if they are properly constructed and are not subjected to irregular pressure, they will require few repairs and will last as long as the mine. Portals to tunnels should be of masonry if the tunnel is to be permanent.

73. Masonry Pillars.—In the Tilly Foster mine, near Brewster, New York, an attempt was made to build artificial

FIG. 90

FIG. 91

pillars in such a manner as to support the hanging wall. These pillars were composed of flat brick arches, which supported a mass of concrete that was intended for the pillar proper. It was discovered that they were not of sufficient strength to carry the hanging wall, and hence they had to be removed and the mine worked as an open pit. After the floors and portions of the ore above these artificial pillars had been removed, they were exposed as shown in Fig. 89, which is reproduced from a photograph.

74. Masonry Arches and Linings.—Arches may consist of either brick or stone. Fig. 90 illustrates a stone arch, which in this case is practically a stull, that has been sprung across a narrow vein to carry the broken material above. Fig. 91 illustrates a similar arch, except that in this case one wall of the vein was so soft that it required lining, and hence the masonry was carried down on one side, as shown in the figure. Fig. 92 shows an elliptic masonry lining; in this case, the car track is supported on timbering in such a manner as to leave the drainage ditch under the track.

FIG. 92

FIG. 93

75. Masonry Walls With Timber or Metal Collars. Where the excavation is not subjected to heavy side pressure but the side walls require some support, the method illustrated in Fig. 93 may be employed. It consists in building straight brick or stone walls at the sides of the tunnel, and using either timber stulls or steel beams across the top of the walls to support the lagging under the roof. The walls may be constructed of brick or stone laid up with cement, or they may be laid up dry with large, firm pieces of rock. If the mine water contains acid or acid salts in solution, the probabilities are that iron or steel beams would be destroyed sooner than wooden beams would rot. There is no reason why steel shapes could not be used to advantage in many ore mines, and their use will undoubtedly increase as timber

decreases. If it is desired to use iron **I** beams or old **T** rails as collars, it would be good policy to place wooden sills from 4 to 6 inches thick on top of the masonry wall and rest the iron on them, for the flanges of iron beams are not wide enough to afford sufficient bearing, and unless the pressure were transmitted over the wall through the medium of some other object, such as timber plates, the wall would crumble. The plate also affords relief in case sudden pressure is thrown on the beam, as wood is more easily compressed than masonry.

76. Iron or Steel Beams.—Iron or steel beams when used to support the roof of excavations must be tightly wedged between the walls and the roof so as to stiffen them and prevent sagging. Plank lagging should be placed above the beam, in order to fill up all spaces between the beam and the roof; but if more filling is needed, round or split lagging may be used above the planks. If split lagging is substituted for plank lagging, the flat side is placed down to afford a wider bearing on the beam, but in this position split lagging is not so strong as when reversed and the flat side is placed in compression and the round side in tension.

77. Brick and Cement Walls With Iron Cross-Beams. An ingenious method of walling a level is advanced by English miners. On each side of the level, which is in a dip mine, brick walls *a*, Fig. 94, are built, and between them are placed walls *b*

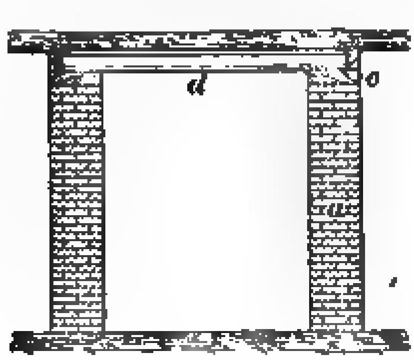


FIG. 94

of rubbish, bound together with cement mortar consisting of four parts of sand and one part of cement. Along the top of the walls so constructed a wooden stringer *c* is placed. If the roof is good, no lagging is used, but the metal beams *d* are tightly wedged against the roof above the walls *a* and *b*.

78. Composite Supports.—One method of supporting the walls and the roof, where the latter is comparatively good, consists in building the cement wall *a*, shown in Fig. 95, at stated distances along the sides of the level.

FIG. 95

These walls are carried to within a few inches of the roof, where a space is left for the sill *b* and the caps *c*. Between the cement walls, brick or stone masonry *d* is built up to about two-thirds the height of the road, and on top of this masonry short pillars of brick *e* or short wooden props *f* are placed to support the stringers. This wall is said to act better than one of all brick and cement, as it deflects to a considerable extent before giving way. Furthermore, it is cheaper than brick masonry, both for material and for the labor required, since unskilled workmen can be employed to erect the cement masonry.

79. Another combination system advanced where the

FIG. 96

roof is good, but where the side walls are liable to fall into the roadway, is shown in Fig. 96. The lower part *a* is a

continuous wall of stone and cement. On this wall at regular intervals brick pillars *b* are erected, and between them short props *c* are placed to support the stringer *d*. This walling and timbering gives general satisfaction, is easy to repair, and when properly constructed will last as long as that part of the mine in which it is placed.

80. Advantages of Composite Supports.—The advantages claimed for brick or stone walls with iron girders instead of using brick arches alone may be stated as follows: Less space is required to be excavated, the saving in this respect being nearly 25 per cent. Less labor and time are required for erection, and therefore the cost is reduced. In soft strata, girders can be placed as the work proceeds; while with brick arching, temporary supports must be used, thus increasing the cost. Girders can easily be removed from one part of a mine to another and used over again; whereas, brickwork can seldom be removed, and is lost when that part of the mine in which it is erected is abandoned.

METAL SUPPORTS

81. Iron or Steel Sets.—Iron or steel beams are best adapted to haulage roads where the pressures are fairly uniform and the strata have settled. Steel girders seldom break when subjected to sudden pressure, but bend in the center, and these deflections may be sufficient to make the area of the haulage road too small; hence, the strata should have settled or the pressures be uniform where iron sets are used, otherwise the roadbed should be made high, as in Fig. 97, so that when settling occurs it may be lowered. Settling of strata is not so liable to occur in roadways where

FIG. 97

pillar-and-room mining is carried on as in other methods of working ore deposits.

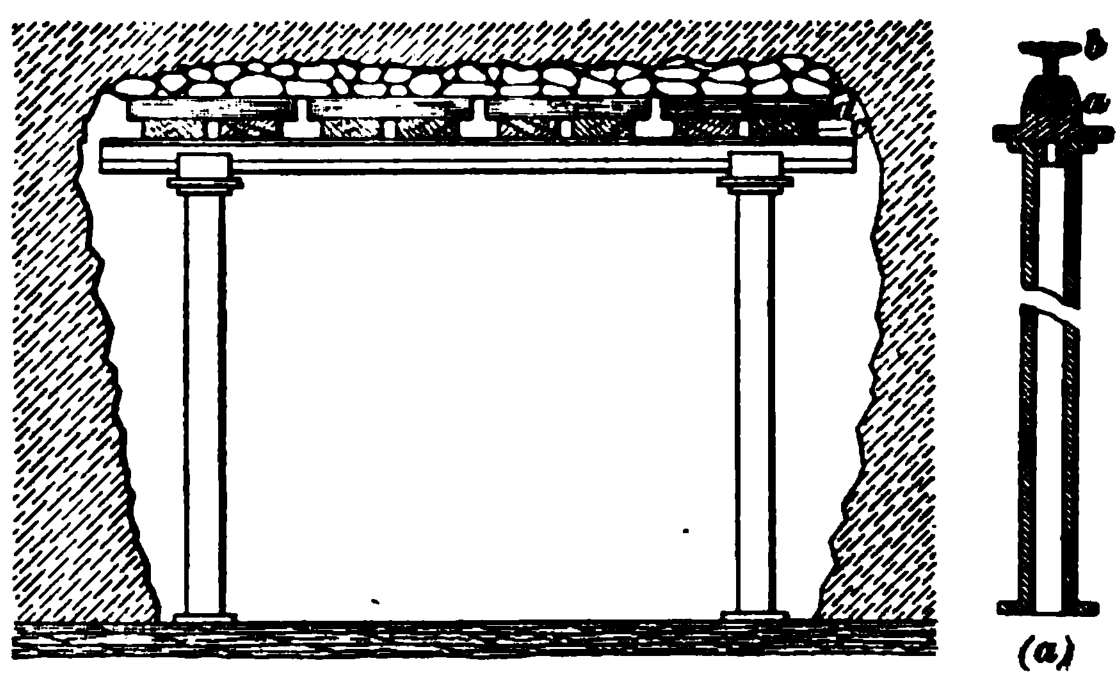
82. Iron and steel I sections weighing from 24 to 60 pounds per yard are bent in the form of a horseshoe; and

FIG. 98

where two sections meet at the top they are jointed by fish-plates, bolts, and nuts. When the roof and the sides are both soft and the floor is subject to creeping, the metal sections are worked to form, as shown in Fig. 98, joined by fish-plates, and then lagged with wood. The sets are placed

about 2 feet apart, and the lagging is driven in as tight as possible. Instead of fish-plates, wrought-iron bands may be placed over the joints and held in place by wooden wedges. In the figure, *c* is bent metal, *b* is the lagging, and *a* is broken stone and cement, for in this particular case the wall was sand. The figure shows a section of the Chicago subway.

83. Cast-iron posts with I-beam caps were employed in a mine in Staffordshire, England. The posts were made hollow, and were flanged at the ends as in Fig. 99 (*a*). A cast-iron chair *a* was made to fit into the post and receive the



(*b*)

FIG. 99

cap *b*. In Fig. 99 (*b*) the chair was made for a 50-pound rail, which was reversed so that the rail flange would rest on the horns of the chair. The lagging *c* was of wood, and above this were arranged planks *d*, in order to wedge up the lagging and make a tight joint. A saving of timber was effected by placing the planks *d* on top of the lagging *c*.

84. Rail Beams.—In some instances, old rails are used as beams when they can be obtained cheaply from the railroads. A method of using them on wooden posts is shown in Fig. 100 (*a*). To prevent the legs from being pushed in at the top, iron bands, made as shown at *a* in Fig. 100 (*b*), are placed in front of the posts shown at *b*, Fig. 100 (*a*).

Wedges *c*, Fig. 100 (*b*), are then driven in tightly, to prevent the bands from slipping and the ends of the rails are wedged as shown at *d*, Fig. 100 (*a*).

(*a*)
FIG. 100

85. Advantages of Iron and Steel Beams.—Iron and steel beams are lighter and easier to handle than wooden beams having the same strength. Moreover, there is no danger of their catching fire; they do not decay; and they give increased space for air to circulate, because their size for a given strength is less than that of timbers.

86. Cost of Iron or Steel Girders.—The cost of I beams varies with the demand, and at the mines the mill price will be increased by the freight rates. The average price of structural beams is about 1.7 cents per pound, although at times it may be as low as $28 a ton. Compared with timber, such beams are twice as expensive; but, on the other hand, they will last from four to six times as long where the conditions are favorable to their use. Owing to varying conditions in different mines, the size of girders must be regulated to the weight that comes on them. Girders should have at least 5 inches web in order to have stiffness, and the weight per linear foot should increase as the span increases.

STRENGTH OF METAL BEAMS

87. Neutral Axis of Metal Beams.—The **neutral axis** is the line passing through a beam on which there is neither compression nor tension. If a load is applied to an iron or steel beam, it will cause the beam to bend a certain amount; and it is impossible for a piece of material to bend without stretching the fibers on the outer side and compressing the fibers on the inner side. Thus, Fig. 101 represents a bar of iron that has been subjected to bending strains; it is compressed at *a* and stretched at *b*. The tension and compression are greatest in those fibers that are farthest from the neutral axis *c d*. Rupture may occur at *b*, but the bar will sustain weight until the rupture has reached the neutral axis, when continued strain will cause it to collapse. The neutral axis passes through the center of gravity of the cross-section of the beam.

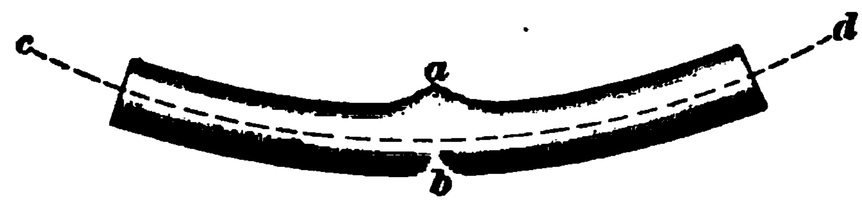


FIG. 101

88. Properties of Metal Beams.—Metal beams are subject to the same natural phenomena as wooden beams. Any load on a beam exerts a destructive force that tends first to bend and next to break it. A beam under the influence of a load exerts two equal and opposite resistances—one to bending and the other to cross-breaking. It is understood, from what has been said concerning wooden beams, that the method of support influences the strength of beams and also the method of loading the beam.

89. Moment of Inertia.—The efficiency of a beam in resisting strains depends on the area and the form of the cross-section of the beam. The effect that form and area have in increasing or decreasing the strength of a beam is termed its **moment of inertia,** *and it may be found by adding together the products obtained by multiplying the area of each part of the cross-section by the square of its distance from the neutral axis.* In most forms of structural material, the moment

of inertia has been obtained by calculus and tabulated, but it may be calculated by dividing a cross-section into squares or triangles and multiplying their areas by the squares of the distance of their centers of gravity from the neutral axis.

The formula that represents the moment of inertia is, for a solid rectangular beam,

$$I = \frac{B D^3}{12}$$

in which B = width of beam;
D = depth.

90. Moments of inertia when referred to the same axis can be added or subtracted. Thus, in the hollow rectangle shown in Fig. 102, if B represents the outside width and D the outside depth, while d represents the inside depth and b the inside width, the moment of inertia of the section can be found by subtracting the moment of inertia of the inner rectangle from the moment of inertia of the outer one, thus forming the equation

$$I = \frac{B D^3 - b d^3}{12}$$

FIG. 102

91. I Beams.—In mining, I beams will be more generally used than rectangular metal beams, and their moment of inertia is found by a similar method of reasoning. A rectangle, Fig. 103, is taken whose height D is the depth of the beam, and whose width B is the same as that of the flanges of the beam; then the rectangle would have an inertia expressed by $I = \frac{B D^3}{12}$. In Fig. 103, there are two rectangles, each equal to d multiplied by b.

FIG. 103

Let t be the thickness of the web; then, $B - t = 2\,b$; and, if t_1 represents the thickness of the flanges, $D - 2\,t_1 = d$. The inertia of the sectional area

to be deducted from the inertia of the rectangle is

$$(B - t)\,(D - 2\,t_1)^3, \text{ or } 2\,b\,d^3$$

hence, the inertia for an I beam is

$$I = \frac{B\,D^3}{12} - \frac{(B - t)\,(D - 2\,t)^3}{12}, \text{ or } I = \frac{B\,D^3 - 2\,b\,d^3}{12}$$

EXAMPLE.—An I beam has 4″ × .425″ flanges and is 8 inches deep; the web of this beam is .27 inch thick. What is its moment of inertia?

SOLUTION.—Here, $B = 4$, $t_1 = .425$, $t = .27$, and $D = 8$. As just shown, $B - t = 2\,b$; hence, $b = \frac{B - t}{2}$. In this case, therefore, $b = \frac{4 - .27}{2} = 1.865$. Also, $d = D - 2\,t_1$, which in this instance is

$$d = 8 - 2 \times .425 = 7.15.$$

Substituting these values in the formula, the following equation is obtained:

$$I = \frac{B\,D^3 - 2\,b\,d^3}{12} = \frac{4 \times 8^3 - 2 \times 1.865 \times 7.15^3}{12} = 57.05+. \quad \text{Ans.}$$

92. Section Modulus.—The **moment of resistance** and the **section modulus** S are one and the same thing. The section modulus is equal to the moment of inertia divided by the extreme distance from the neutral axis, multiplied by the modulus of rupture, or the fiber stress. If T represents the distance from the neutral axis to the farthest fiber of a beam, the section modulus will be represented by $\frac{I}{T}$, in which T is taken in inches, and is the distance of the center of gravity of a section from the top or from the bottom.

Since the moment of inertia is expressed by the formulas $I = \frac{B\,D^3}{12}$ and $T = \frac{D}{2}$, then,

$$S = \frac{\frac{B\,D^3}{12}}{\frac{D}{2}} = \frac{B\,D^3}{12} \times \frac{2}{D} = \frac{B\,D^2}{6} \times \text{modulus of rupture}$$

93. Properties of Standard I Beams.—Table I gives the United States Steel Company's standard sizes of I beams. The headings are descriptive of each column. The different values given in column 2 are the results due

to having different thicknesses of web for beams of the same depth, and different widths of the flanges.

94. The **modulus of rupture** and the **fiber stress** are synonymous terms, and mean the safe strength of a beam. The safe loads for **I** beams are taken at 7,000 pounds per square inch of section for cast iron; 12,000 pounds per square inch of section for wrought iron; and 16,000 pounds per square inch of section for steel beams. For riveted steel girders, the safe load is generally taken at 13,000 pounds. As the strength of a beam is inversely as its span, the safe load for any span may be obtained by dividing the safe load for 1 foot of span by the span in feet.

Let W = total load on beam, in pounds;
L = span, in feet;
C = coefficient of fiber stress given, in pounds or tons.

Then, the values for the safe load x, in pounds, and the value for section modulus S may be found for beams supported at both ends and loaded in the middle by the following formula:

$$x = \frac{S\,C}{3\,L}, \text{ or } W = \frac{S\,C}{3\,L}; \text{ hence, } S = \frac{3\,L\,W}{C}$$

EXAMPLE.—A steel I beam of 20 feet span has to support a load at its middle of 24,000 pounds; what must be the size and weight of the beam?

SOLUTION.—Here, $L = 20$; $W = 24{,}000$; and $C = 16{,}000$; then, from the formula,

$$S = \frac{3 \times 20 \times 24{,}000}{16{,}000} = 90$$

In column 6 of Table I, 93.5 is found to be the nearest section modulus, which is the factor for a 60-lb. 18″ × 6″ beam with a web .55 in. thick. Ans.

95. The safe load for metal beams supported at both ends, with the load uniformly distributed, is found from the following formula:

$$x = \frac{2 \times C}{3\,L}, \text{ or } W = \frac{2 \times C}{3\,L}; \text{ hence, } S = \frac{3\,L\,W}{2\,C}$$

TABLE I

PROPERTIES OF STANDARD I BEAMS

1	2	3	4	5	6
Depth of Beam Inches	Weight per Foot Pounds	Area of Section Square Inches	Thickness of Web Inch	Width of Flange Inches	Section Modulus (S). Neutral Axis Perpendicular to Web at Center
24	100.00	29.41	.754	7.254	198.4
24	95.00	27.94	.692	7.192	192.5
24	90.00	26.47	.631	7.131	186.6
24	85.00	25.00	.670	7.070	180.7
24	80.00	23.32	.500	7.000	174.0
20	75.00	22.06	.649	6.399	126.9
20	70.00	20.59	.575	6.325	122.0
20	65.00	19.08	.500	6.250	117.0
18	70.00	20.59	.719	6.259	102.4
18	65.00	19.12	.637	6.177	97.9
18	60.00	17.65	.555	6.095	93.5
18	55.00	15.93	.460	6.000	88.4
15	55.00	16.18	.656	5.746	68.1
15	50.00	14.71	.558	5.648	64.5
15	45.00	13.24	.460	5.550	60.8
15	42.00	12.48	.410	5.500	58.9
12	35.00	10.29	.436	5.086	38.0
12	31.50	9.26	.350	5.000	36.0
10	40.00	11.76	.749	5.099	31.7
10	35.00	10.29	.602	4.952	29.3
10	30.00	8.82	.455	4.805	26.8
10	25.00	7.37	.310	4.660	24.4

TABLE I—(*Continued*)

1	2	3	4	5	6
Depth of Beam Inches	Weight per Foot Pounds	Area of Section Square Inches	Thickness of Web Inch	Width of Flange Inches	Section Modulus (*S*). Neutral Axis Perpendicular to Web at Center
9	35.00	10.29	.732	4.772	24.8
9	30.00	8.82	.569	4.609	22.6
9	25.00	7.35	.406	4.446	20.4
9	21.00	6.31	.290	4.330	18.9
8	25.50	7.50	.541	4.271	17.1
8	23.00	6.76	.449	4.179	16.1
8	20.50	6.03	.357	4.087	15.1
8	18.00	5.33	.270	4.000	14.2
7	20.00	5.88	.458	3.868	12.2
7	17.50	5.15	.353	3.763	11.2
7	15.00	4.42	.250	3.660	10.4
6	17.25	5.07	.475	3.575	8.7
6	14.75	4.34	.352	3.452	8.0
6	12.25	3.61	.230	3.330	7.3
5	14.75	4.34	.504	3.294	6.1
5	12.25	3.60	.357	3.147	5.4
5	9.75	2.87	.210	3.000	4.8
4	110.50	3.09	.410	2.880	3.6
4	9.50	2.79	.337	2.807	3.4
4	8.50	2.50	.263	2.733	3.2
4	7.50	2.21	.190	2.660	3.0
3	7.00	2.21	.361	2.521	1.9
3	6.50	1.91	.263	2.423	1.8
3	5.50	1.63	.170	2.330	1.7

EXAMPLE.—A steel beam of 20 feet span has to support a uniformly distributed load weighing 24,000 pounds; what must be the dimensions of such a beam?

SOLUTION.—Here, $L = 20$; $W = 24{,}000$; and $C = 16{,}000$; then, from the formula just stated,

$$S = \frac{3 \times 20 \times 24{,}000}{2 \times 16{,}000} = 45$$

In column 6, Table I, the nearest number to 45 is 58.9, and this section modulus calls for a 15″ × 5.5″ beam weighing 42 lb. per ft., and having a web .41 in. thick. Ans.

96. Iron Props.—Fig. 104 is a hollow iron prop made in two sections *a* and *b* with a sleeve *c*. This prop has been used in long-wall working where it is necessary to draw the prop to let the roof sag. By knocking up the sleeve *c*, the prop falls down and can be pulled out of danger. In case one or the other section of the prop is buried by rock, it can usually be recovered by a chain attached to the parts *a* and *b*. This chain is strong enough to recover the end from under the first fall of rock.

FIG. 104

FIG. 105

Fig. 105 (*a*) shows the section of an I beam that has its web *a* cut down so that the flanges *b* can be turned down over it, as in Fig. 105 (*b*), to form the ends of a prop. As these are more expensive than wooden props, holes *c* are punched in the web, and in these a hook can be inserted to help withdraw the beams for future use. Cast-iron props are heavy, and are liable to become broken when pulled; hence, this prop, being lighter and tougher, is preferred. This I prop is patented, and is known in England as *Firth's prop*.

ASSAYING

(PART 1)

QUANTITATIVE ANALYSIS

INTRODUCTION

1. The determination of the quantity of an element in any given substance is termed **quantitative analysis.** Assaying is the quantitative analysis of ores, minerals, and metallurgical products, and is practiced in order to ascertain their value.

There are two general ways of making analyses, the wet way and the dry way. Wet assays are generally spoken of as *analyses*, to distinguish them from dry, or fire, assays, although the term *assay* is equally applicable to both. The two methods of assaying differ so radically that, in order to avoid confusion, they will be described separately. With the exception of gold and silver, all the metals can be more accurately determined in the wet way than in the dry way. This practically restricts fire-assaying to the determination of gold, silver, and lead. In America, lead ores are almost entirely bought and sold on the results obtained by fire-assay; owing to the fact that a certain percentage of the metal is volatilized in smelting operations. The fire-assay of lead is performed in such a way as to correspond closely with the conditions that prevail during the smelting of lead ores for base bullion. For the exact chemical analysis of lead, the

wet assay is to be employed. Copper and tin ores were once assayed by fusion, but this method has been almost entirely superseded by wet assaying, which in the case of copper is much more accurate.

FIRE-ASSAYING

PRELIMINARY OPERATIONS

2. The work and apparatus of fire-assaying will be described as nearly as possible in the order of the actual operations.

The outfit of an assay office will naturally vary somewhat, depending not only on the amount and character of the work to be done, but also on the individual preferences of the assayer. Only the most essential apparatus of the fire-assay office are given here. Many of the articles listed by dealers in assay supplies, though convenient in special cases, are of little or no use in the ordinary run of fire-assaying. There are, however, some labor-saving devices that the assayer will find very convenient to adopt, as they will lessen his work; such articles and their uses will be taken up in order.

3. Samples for Assaying.—As a rule, the samples sent to the public assay office are less than 1 pound in weight, but at mills and smelters the samples weigh several pounds. The sample should never be so small that there will be insufficient ore left to make a duplicate assay, for an accident may ruin the first assay, or doubt may be raised as to its accuracy. Samples for assaying are so finely pulverized as to pass through an 80 or 100 screen—that is, a screen having 80 or 100 openings to the linear inch, or 6,400 to 10,000 openings to the square inch. When the material is finely pulverized, it goes more rapidly into solution, particularly if it is of a refractory nature. To prepare a sample for assay, apparatus and tools are required, and these are described in the order of their use.

4. Crushers.—When the ore comes to the assayer in lumps, it is broken up into pieces not more than $1\frac{1}{2}$ inches thick. This coarse crushing may be accomplished with a hammer, or in a large iron mortar if a specially made bar is used as a pestle. A 6-quart mortar will usually be large enough, although mortars are made to hold as much as 16 quarts. Fig. 1 shows a cast-iron mortar with a wide pedestal to prevent it from upsetting; a cast-iron pestle is also shown for breaking up the pieces of rock to a size suitable for the crusher or to a size less than $1\frac{1}{2}$ inches in diameter. Cast-iron pestles are usually made of brittle white iron, and are therefore easily broken, for which reason steel or wrought-iron bars with knobs on one end are much more durable and should be adopted. The bar should be about 3 feet long, so that the operator may stand upright while crushing. After breaking the ore in the mortar, it is transferred to a small jaw crusher, similar to that shown in section in Fig. 2, which reduces it in size until no piece is larger than $\frac{1}{4}$ inch thick. Where the samples to be reduced are large and numerous, the laboratory crusher is driven by a small motor; but in an ordinary laboratory, it is turned by hand. The material that passes through the crusher is caught in a suitable sheet-iron pan, and transferred to a sampling cloth.

FIG. 1

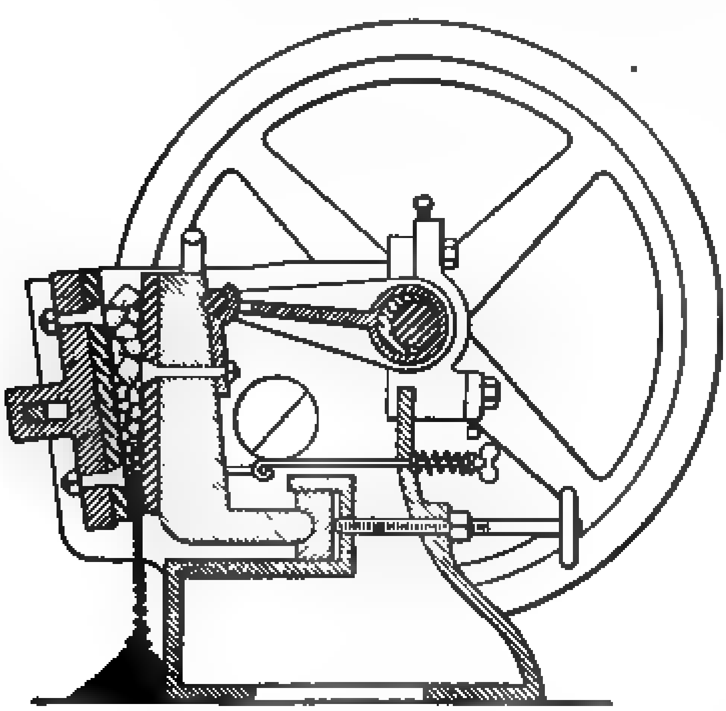


FIG. 2

5. Samplers.—The ore is thoroughly mixed on the sampling cloth and is then shoveled by the tin sampling scoop *a*, Fig. 3, on to the sample riffles *b*, where half the ore is caught, while the other half passes between the riffles. The

FIG. 3

ore is spread evenly over the riffles, and not heaped up on them in any one place. The sample taken may be that which is caught in the riffles, or that which passes between them. The sampler in Fig. 4 is an improvement on the tin sampler described, since all the material passes from the scoop through riffles *a* to pans *b* and *c*, thus giving duplicate samples, which may be worked down, and one assayed against the other as a check. In assaying, very much depends on the care taken in sampling; when the results obtained by duplicate assays of the same sample do not agree, there is something radically wrong, which necessitates check assaying. In case the sample is large, it may be worked down by a mechanical sampler, such as that shown in Fig. 5. The ore is placed in the hopper *a* of this machine, and falls into the bucket *b*, thence into *c*, and so on to the discard bucket *d*. The buckets *b* and *c* are divided into four compartments, two of

FIG. 4

which have bottoms, while the other two are open. The sample is halved in the upper bucket, and again halved in the lower, thus allowing one-quarter of the original sample to reach the discard bucket. Refeeding the quarter obtained would furnish a sample of one-sixteenth the quantity of the original sample, and a second refeeding would reduce it to one sixty-fourth of the original sample. The machine is operated by a crank *e*, which when revolved works a cam that slightly tips the hopper by striking a finger *f*, connected with the hopper by the spring *g*. The buckets rotate in opposite directions, thus taking the whole stream part of the time and not a part of the stream the whole time. Mechanical samplers for laboratory work are driven by power, where the ore to be sampled—as at smelters, mills, and custom samplers—is received in large quantities, and by hand, in private laboratories.

6. Quartering.—The method of reducing samples by **quartering** takes its name from the peculiar manner in which it is performed. Suppose an assay sample is to be taken from a 100-pound sample of ore, made up of coarse and fine lumps, and representing, perhaps, a carload of ore. The sampling floor is first swept thoroughly, to prevent any rich dust from former samplings becoming mixed with the sample in hand. Then, the entire sample is run through a

Fig. 5

crusher, which reduces the lumps to a maximum diameter of, say, 1 inch. If the ore is high grade or contains rich lumps in a comparatively barren gangue, the first crushing should be considerably finer, in order to make the even distribution of the values more certain. Two or three 1-inch lumps more of rich ore in one half of a sample than in the other, when the main mass of the sample is barren gangue material, would cause quite a serious error, but, if the ore is crushed to $\frac{1}{4}$ inch as the maximum diameter, a lump more or less of rich ore on either side would cause an error only one sixty-fourth as great as that from a 1-inch lump, as the volume is in proportion to the cube of the diameter, and the chances of the ore being evenly mixed will also be much greater.

FIG. 6

The crushed ore is now thoroughly mixed on the floor and heaped up into a conical pile, each shovelful being carefully upset on the apex of the cone, so as to run down all around. The sampler now walks round and round the heap, continually raking down a little ore from the pile with the shovel, until the ore is in a flat, circular pile, Fig. 6, about 4 inches thick. The pile is then marked off with a stick into quadrants, as shown in the figure by the two diametrical lines at right angles to each other. Two alternate quarters are then carefully shoveled away. Thus, considering the quadrants as numbered consecutively from *1* to *4*, Nos. *1* and *3* would be discarded, and Nos. *2* and *4* saved, or vice versa. This leaves but half the sample to be operated on; and this, if the work has been carefully done, should be of the same value as the discarded portion. The two quarters that were saved are again crushed—this time to a maximum diameter of perhaps $\frac{1}{8}$ inch—and the entire operation is repeated. This will leave a sample only one-fourth as large as the original sample, but of the same average value. The

sample is further reduced by continued quartering, or by using the tin sampler, to about $\frac{1}{4}$ pound; and this is pulverized and screened.

7. For mixing small samples, say 10 pounds and less, heaping up and shoveling will be found a rather tedious and clumsy process. After the sample is down to that size, and for small assay samples, the following method is generally adopted:

The crushed sample is placed on a sheet of oilcloth or rubber cloth, which is placed on the floor or table; the alternate corners of the cloth are then drawn over, one at a time, toward the corners diagonally opposite, rolling and mixing the sample within. When thoroughly mixed, the sample is heaped up into a conical pile by drawing the corners of the cloth upwards. The pile is then flattened as follows: A thin sheet of iron, held as shown in Fig. 7, is pressed down slightly into the apex of the cone and twisted gently around. As the plate revolves, it flattens and spreads out the ore, the process being continued until the pile is reduced to the desired thickness. The pile is then marked out into quadrants with a spatula, and quartered as already described. Quartering has been found by experiment to give accurate average samples, if the work is carefully done.

Fig. 7

Fig. 8

8. Drying the Sample.—Before pulverizing, wet samples must be dried on a water bath, Fig. 8, or in a drying

oven, Fig. 9. Clayey ores will not pulverize readily until the moisture has been expelled, and this cannot readily be accomplished below a temperature of 100° C., or 212° F. Iron and manganese ores sometimes contain considerable moisture, and are purchased on an analysis of dry ore. Suppose, for example, an iron ore contained 5 per cent. of moisture; then, in 2,000 pounds there would be 100 pounds of water, and the purchaser would pay one-twentieth of the price of the ore for water.

FIG. 9

In use, the water bath is placed above a suitable heater, as also is the dryer. The walls of the water bath are double and contain water, thus preventing the temperature rising above 100° C. The spout shown is for the escape of steam. The air dryer has an opening *a*, in which is inserted a thermometer, by means of which the heat is recorded and regulated.

9. Bucking Board.—After the sample has been reduced to 1 pound or less, it is placed on a cast-iron plate, Fig. 10, termed a **bucking board** and pulverized by the cast-iron **muller** *a*. Mullers with handles weigh from 12 to

FIG. 10

38 pounds; one weighing about 18 pounds is sufficiently heavy for laboratory work. In case there are lumps of ore as large as a pea, the ore should be sifted and the coarse lumps pulverized in a small iron mortar and then returned to the bucking board. The muller is moved back and forth with the right hand, while the left hand rests on the muller. The sample is pulverized until it will pass through an 80- or 100-mesh sieve. The operator has a brush, with which he sweeps the pulverized material into the trough *b*, and from this trough into a sieve. The material that will not pass through the sieve is returned to the bucking board and is reground until it does. After a sample has been ground, the bucking board must be thoroughly cleaned to prevent **salting,** that is, increasing the quantity of gold in the next sample bucked.

Bucking boards and mullers can be obtained with rough or planed surfaces. While the rough boards and mullers are cheaper and eventually wear smooth, they are difficult to clean – hence the planed boards are to be preferred. There are other kinds of bucking boards that may be obtained; one of these is circular, and still another is suspended on trunnions at the sides; the one shown in Fig. 10 is, however, the standard.

FIG. 11

10. Buck's Mortar.—For the purpose of pulverizing a sample of ore, without the arduous work required for bucking, the machine shown in Fig. 11 was invented. The mortar has a spindle *a* in the center of the bowl, around which the muller *b* is rotated. The handle *c* is for the purpose of rotating the muller horizontally around the pivot, and lifting it from the mortar. As the muller weighs 28 pounds and the mortar 32 pounds, there is little saving in work, for both must be lifted after grinding and regrinding. For grinding a sample with quicksilver to amalgamate the gold, or for the

purpose of grinding amalgam, Buck's mortar will be found serviceable; it cannot, however, replace the bucking board and muller as a sample pulverizer.

11. Sieves.—The cloth for fine screening is usually woven brass wire. The opening between the wires is called a *mesh*, and the number of meshes in a linear inch, designates the number of the screen; thus, a No. 60 screen is a screen that has 60 openings to the inch or 3,600 openings to the square inch. There are not always this number of holes in a square inch; but the size of the holes is such that there would be this number if the wire could be made of sufficiently small diameter to retain economical strength. Fig. 12 (*a*) shows a tin sieve *a* fitted to a pan *b*. This arrangement saves time and labor, and also catches fine *pulp*, as pulverized ore is called. Fig. 12 (*b*) shows a nest of sieves, with the size marked on each sieve. There is also a cover *a* to retain the impalpable pulp, and a pan *b* to use in connection with the sieves.

(a)

(b)

Fig. 12

The price of sieves increases with their fineness; for rough work it is customary to use a wooden-frame sieve with iron wire. After a sample has been sifted, the sieve must be turned upside down and lightly whipped and brushed, in order to remove ore or gold scales—otherwise the following sample may become salted. The ore sample from which the assay sample is taken is all put through the sieve. If metallic scales are left on the sieve they are tested with a magnet to see if they are iron, and if not they should be collected and assayed separately. Some assayers prefer to cover the scales with fine ore and grind them until they pass through the sieve, rather than go to the trouble of

making a separate assay and the necessary calculations. The better method, unless the scales are silver, is to assay them separately, and add their value to the assay value of the ore that passed through the screen.

12. Dividing the Pulp.—After the sample has passed through the sieve, it is transferred from the pan to a piece of oilcloth, and thoroughly mixed by drawing up first one corner of the oilcloth and then another, in order to roll the material over. After the ore is mixed, it is spread out in a thin layer and small portions of the pulp are taken with a spatula from various parts of the layer, as shown at *a*, Fig. 13, and placed in a dish *b*. Other parts of the pulp are

FIG. 13

taken in the same manner and placed in another dish. The assays are run in duplicate, one charge of ore being taken from each dish. Some assayers prefer to mix the sample, pour it in a dish, and weigh the pulp for assay from this dish. The dish containing the pulp must not be tapped or disturbed or allowed to stand long, as there is danger of the heavier particles of ore sinking through the pulp to the bottom of the dish. If the ore is not immediately weighed, the sample must be remixed on the oilcloth.

13. Spatulas.—The steel **spatula** is a form of knife used in the laboratory for mixing or sampling pulverized ore

or for handling other materials about the laboratory. Fig. 14 shows a common form of spatula. The blade should have some spring and at the same time should be stiff enough so that it will not be broken if used for digging material out of a bottle or for similar purposes. The laboratory should usually be provided with several sizes of spatulas, from those having a 3- or 4-inch blade for use at the balances, etc., to those having 10- or 12-inch blades for use in mixing several pounds of material and for sampling large pulps, etc.

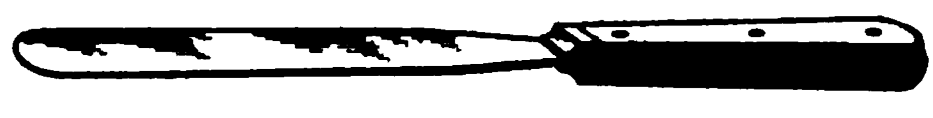


FIG. 14

Horn spatulas are frequently employed for removing precipitates from beakers, and for similar purposes where steel spatulas would be attacked by the acids in the solution.

14. Weighing the Pulp and Assay.—The result of a gold-silver assay or any analytical determination is dependent on the accuracy with which the ore to be assayed (termed the *ore charge*) is weighed, and, more particularly, the button resulting from the assay. No matter how well the rest of the work is done, an error in weighing either the charge or the button will render the result, as a quantitative analysis, worthless, and the assayer's work will be futile, as the amounts to be weighed are so small that a very slight error will multiply itself enormously when the results are reduced to the basis of ounces of metal in a ton of ore. The extreme delicacy required in assay balances and the care required in weighing will be better realized when it is understood that scales are made so sensitive that they will indicate $\frac{1}{200}$ milligram, or .00007715 of a troy grain. Ores containing less than $1 per ton in gold are in some instances worked profitably, and much smaller proportions of gold than this can be recovered by fire-assay and accurately weighed.

15. Assay Ton.—As ore is weighed by avoirdupois weight, and gold by troy weight, the assay-ton system has been devised to simplify matters. An avoirdupois pound contains 7,000 grains troy weight; hence, a ton of 2,000

pounds contains 14,000,000 grains troy weight. As 1 troy ounce contains 480 troy grains, there are therefore $\frac{14{,}000{,}000}{480}$ $= 29{,}166 +$ troy ounces in 2,000 pounds avoirdupois. If it is assumed that 1 mg. represents 1 ounce troy, then, by taking as many milligrams as there are troy ounces in 1 ton avoirdupois, the **assay-ton** system is originated.

An assay ton, written **A. T.**, contains 29,166+ mg., and each milligram is equivalent to 1 troy ounce; therefore, if an assay button from an assay ton of ore weighs 5 mg., it indicates that there are 5 troy ounces of gold in an avoirdupois ton. This is expressed better by the proportion:

2,000 lb. av. : 1 oz. troy = 1 A. T. troy : 1 mg.

16. If more or less than 1 A. T. of ore is used, the contents of the ore in ounces per ton may be found by dividing the weight of the button, in milligrams, by the weight of ore taken, in assay tons. For example, if 2 A. T. of ore are taken, the resulting button would be twice as heavy as the button from a 1-A.-T. charge; and its weight would, therefore, have to be divided by 2 to obtain the weight of the button from 1 A. T., in which each milligram represents 1 troy ounce per ton. With a $\frac{1}{10}$-A.-T. charge of ore, the button is only one-tenth as large as the button from 1 A. T.; consequently, each milligram represents ten times as much as the same weight in the button from a full A.-T. charge; or, in other words, the weight of the button from $\frac{1}{10}$ A. T. of ore multiplied by 10 is equal to the weight of the button from 1 A. T. The following general rule, therefore, may be adopted for the calculation of the results from the assay of *any* weight of ore:

The weight of the button in milligrams divided by the weight of ore taken in assay tons gives the number of ounces per ton.

If the weighing is done with grain weights, the assay ton may be assumed to contain 29,166+ grains, and 1 grain of gold or silver extracted will then represent 1 ounce troy to the ton of ore. (For tables of metric weights and assay-ton weights, see pages 47 and 48 of this Section.)

17. Weights.—For weighing assay pulp, the metric system of weights has been generally adopted by assayers, as it greatly simplifies the calculations, all the weights being divided decimally. The assayer should have two sets of weights—one set of metric weights, from 20 or 50 g. to 1 mg., and one set of A.-T. (assay-ton) weights, from 4 A. T. to $\frac{1}{20}$ A. T. An accurate set of metric weights as just described costs from $9 to $14. The set usually includes three riders. A set of assay-ton weights as described costs about $6.

The metric weights come in a box, and the fractions of a gram are made of platinum except those below 20 mg., which are made of aluminum. The weights above the gram are made of brass, and are lacquered to prevent corrosion. There are spaces in the box for each weight, and also for the riders and pincers.

The weights should always be handled with the pincers supplied for this purpose, as the moisture of the hands corrodes them, and may also appreciably alter the weight of smaller weights. They should always be returned to the box as soon as the weigher is through with them, both to prevent their loss and to save time and prevent errors from the overlooking of small weights that may have been left in the pan from a previous weighing.

ASSAY BALANCES

18. Pulp Balance.—The balance for weighing the pulp should have a capacity of at least 200 g. in each pan, and should be readily sensitive to a difference of 5 mg. in the weights in the two pans. A convenient pulp balance is shown in Fig. 15. Such a balance costs, together with a glass case, about $25. Pulp balances are frequently used without glass cases, and thus the cost is reduced somewhat. Two accurately balanced watch glasses make the best possible removable scale pans, though metal pans are frequently used. The watch glasses are made with a glass lip or handle for convenience in removing.

19. Button Balance.—The balance for weighing the gold and silver buttons should have a capacity of 1 g. or 2 g. in each pan, and should be sensitive to $\frac{1}{50}$ mg. or less. Balances are made that indicate a variation of $\frac{1}{500}$ mg. Such extreme accuracy as this is, however, unnecessary in ordinary work, and such balances are expensive, ranging in price from $125 to $250. Balances sensible to $\frac{1}{20}$ mg. can be bought for from $65 to $80, and for $90 or $95 a balance may be obtained sensible to $\frac{1}{100}$ mg.

Fig. 16 shows a button balance with rider attachment, with which device all the more delicate and higher-priced balances are equipped. The "rider" is a small loop of platinum or aluminum wire, of the shape shown in Fig. 17, and of definite weight (usually 1 mg. for button balances and 10 mg. and 12 mg. for less delicate analytical balances). This rider is set astride the beam of the balance, which is graduated like the beam of a steelyard. The two pans are brought nearly to a balance by the use of ordinary weights, and the rider is then moved along the beam until the balance between the two sides is perfect. Each division on the beam of the balance is equivalent to a certain fraction (usually $\frac{1}{10}$, $\frac{1}{20}$, or $\frac{1}{50}$) of a milligram. Thus, if each division is one-fiftieth of the length of the beam from the middle bearing to the end, or pan bearing, and a 1-mg. rider is used, each division that the rider is distant from the middle bearing is equivalent to a weight of $\frac{1}{50}$ mg., or .02 mg. in the pan on the same side; that is, it will balance that weight in the pan on the other side. If a 10-mg. rider is used, each division represents $\frac{10}{50}$, or $\frac{1}{5}$, mg., or .2 mg. The weight thus indicated by the rider should be added to the sum of the weights in the pan on the same side, to obtain the weight of the button in the opposite pan. If the rider is used on the opposite side from the weights, the amount indicated by the rider must be subtracted from the sum of the weights, in order to get the correct weight of the charge.

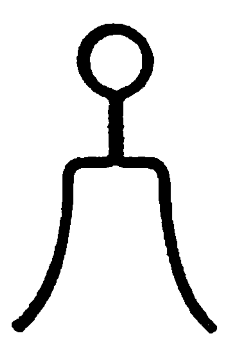


FIG. 17

The rider is moved back and forth along the beam by a hook on the end of a sliding rod extending through the side of the case of the balance. This enables the assayer to do the final balancing with the case closed, so that the balance cannot be disturbed by drafts. The rider is extremely convenient, as it does away with the use of very small weights, and renders the accurate adjustment of the balance much more rapid and easy.

20. Analytical Balance.—For weighing material used in gravimetric analysis, and for weighing precipitates,

platinum or porcelain crucibles, and similar objects, which require the handling of material too heavy for the button balances, an analytical balance is necessary. This is constructed considerably like the balance illustrated in Fig. 15, but is provided with a rider and should have a capacity of from 100 g. to 200 g. in each pan and be sensitive to a weight of from $\frac{1}{5}$ mg. to $\frac{1}{10}$ mg. Such balances can be obtained for from $50 to $75 each. In some laboratories, analytical balances are used for weighing the pulp, while balances such as shown in Fig. 15 without the glass cover are used for weighing out the fluxes.

Directions for setting up are furnished with each set of balances, and these should be carefully observed, as balances are very delicate instruments and are easily injured. The button balance, in particular, should be set in a dry place, away from the furnace, so that it will not be affected by the heat. Any sudden change of temperature is bad; and even the shifting of the sunlight, if it strikes the balance, is sufficient to throw it out of adjustment. No jar or disturbance of any kind should be permitted, as it not only interferes with the weighing, but injures the balance. In ordinary buildings it is difficult to secure a solid support, for if the table on which the balance is set rests directly on the floor, the vibrations from persons walking around the room will be a serious annoyance. In such a case, it is a good plan to set the support for the balances on wooden posts or brick piers, set in the ground underneath the office and projecting upwards through the floor without touching it. The balances should always be tested before using, to see that they are in perfect adjustment.

The beam of a balance, while weighing, is supported on steel or agate knife edges, and the pans are also hung from knife edges, thus making the balance almost frictionless. When not in use or while charging, the beam is raised from the knife edges by turning a knob projecting through the front of the case, and the weight of the pans is supported from beneath by rests or stops, which are worked either by a separate knob or, in most button balances, by the same

knob that releases the beam. These rests should always be raised while putting on or taking off weight from either pan, as the knife edges will be dulled if the weight is allowed to fall on them while charging.

To protect balances from moisture, a small beaker, partly filled with strong sulphuric acid, may be placed in the case. The acid will absorb the moisture from the air in the case, and thus prevent rusting of the balance. A small quantity of calcium chloride, $CaCl_2$, in a glass desiccator, Fig. 18, placed inside the balance case, will answer the same purpose as the sulphuric acid, and will not cause so much trouble if accidentally upset.

Fig. 18

METHOD OF WEIGHING

21. Before starting to weigh, it should always be ascertained whether the balances are in perfect adjustment. They must, first of all, be perfectly level. Leveling screws and bubble tubes are provided for this adjustment. The scales are then brought to a perfect balance by means of the adjusting appliance on the beam or by the use of the rider. They are now ready for weighing.

When weighing with delicate balances, or with practically any balance for analytical work, it is, as a rule, best to weigh on the "swing," as it is expressed, rather than to attempt to bring the scale to perfect rest. For instance, if the weights have been placed in one pan and the load in the other, the supports are carefully removed and the pans allowed to swing slightly. The long pointer will travel backwards and forwards across the graduated scale between the pans. If after the first two or three swings the pointer were to travel seven divisions to the left, six and one-half divisions to the right, six divisions to the left, five and one-half divisions to the right, etc., thus dropping off a half division (or other equal

fraction of a division) each time, but swinging approximately equal, the pans would be equally balanced and the weight would be read without spending the time necessary to bring the beam to rest. The adjustment of the scales should be made in the same way; that is, they should be made to swing to equal distances on both sides of the center, either when the pans are empty or when they contain equal weights. Usually, in a good balance, pans will not come to rest as rapidly as a half division for each swing, and hence it might require a few minutes to bring them to rest with the pointer in the center. This method of weighing by means of the deflection of the needle to the right and left is both much more rapid and accurate than the system of trying to bring the pans to rest, but at the same time the swing should not exceed a deflection of more than four or five divisions each way.

22. Charge.—The weight of the charge is usually fixed, and so the desired weight is put into one pan. The pulp from which the charge is to be taken is poured on a sheet of glazed paper or a mixing cloth (a sheet of rubber cloth or oilcloth, about 10 in. × 15 in.) in front of the scales, thoroughly mixed by rolling, as described in Art. **6**, and then with a large spatula spread out into a thin pile. The empty scale pan is then taken out and the pulp charge placed in it. The charge is taken from all over the surface of the pile of pulp, a dip here and a dip there, to further insure an average sample, as explained in Art. **7**. When approximately the right amount is in the pan, it is replaced on the scales. Then pulp can be added or removed as required with a small spatula until the correct weight is obtained. The beam is held off the knife edge during charging, being let down only to observe the balance, until very nearly the correct weight is struck. It may then be let down; but the pan rests are still kept up, the pans being released only for observing the balance, and then immediately stopped again. As soon as the correct weight is obtained, the beam is raised from the knife edge, and the pan removed and its contents brushed into the crucible or

other receiving vessel, using a soft camel's-hair brush for this purpose. If duplicate charges are being used, the charges may be weighed separately, or the weights may be removed before emptying the loaded pan, and a charge put into the pan that contained the weights, to balance the weighed charge. The pulp should be rolled and mixed anew for each charge and taken from the second dish mentioned in Art. **12** for a duplicate assay. The pans must always be brushed perfectly clean after each weighing, and should be handled as lightly as possible.

One objection to weighing by balancing charges is that there is danger of the pans of the balance being interchanged, and, as a rule, the pans are not exactly of the same weight, and hence should be kept in their respective places. Most chemists make it a rule to do all their weighing in one pan and always to use the weights in the other. One advantage of this system is that the small weights are never in danger of becoming mixed in with the ore, or there is no danger of ore or pulp becoming attached to the pan in which the weights are used. When the charge is always weighed in one pan, a right-handed man will usually find it most convenient to place the weights in the right-hand pan.

23. Buttons.—The buttons are weighed in practically the same manner as the pulp, except that the unknown weight (button) is put in one pan and then balanced by weights in the other pan. When the weights are always used in one pan, a right-handed man will find it convenient to place the weights in the right-hand pan. The final balance is usually obtained by the use of the rider. If duplicate buttons are being weighed, the weight of one may be obtained as described, and then the second button substituted for the weights in the weight pan, and the difference in the weights of the buttons, if there is any, balanced by the use of the rider (and the small weights, if necessary, although a difference great enough to require as much as a milligram to measure it is altogether too large to overlook, except in extremely high-grade gold ores or rich silver ores). The

front of the scale case should be closed as soon as the difference in weight comes within the limit of the rider, and the final balance obtained without any interfering air-currents. The work can be done about as quickly by leaving the weights in the pan and substituting one button for the other and noting the difference with the rider.

The assayer should have a weight book in which to record all weighings of buttons or other materials obtained in assaying or analytical work; and in reading any given weight he should first read the weight as recorded by the weights in the pan of the balance and the setting of the rider, and then should read the weight as recorded by the empty spaces in the weight box and the rider. This will give a check on the reading and will also serve to check the loss or oversight of any small weight on the scales.

FURNACES

24. The furnaces used in assaying vary in size and design with the number of assays to be made and the nature of the work done, and also with the kind of fuel used. They are of two general types: *wind*, or *crucible*, *furnaces*, in which the vessel containing the charge to be melted is heated by direct contact with the fuel or flame, and *muffle furnaces*, in which the heating is indirect, the vessel being placed in a fireclay muffle or oven, which is heated from the outside. Charcoal, coke, coal, gas, or gasoline may be used as fuel with either type.

25. Wind Furnace.—Wind furnaces are used chiefly for melting bullion, for which purpose a higher heat is necessary than can be obtained in a muffle furnace; the crucibles used for bullion melting are, moreover, too large to enter the ordinary muffle. Occasionally, the wind furnace is used for making fusions of larger quantities of ore than can be safely put into crucibles of a size not too large for the muffle furnace. The American practice, however, is to use the muffle furnace wherever possible, as it is cleaner and

neater, and the work can be more carefully watched and the heat and air more successfully controlled.

Fig. 19 shows the ordinary type of wind furnace for burning solid fuel (anthracite, coke, or charcoal). The furnace is built of red brick, with a firebrick lining at least 4 inches thick in the fire-chamber, the entire thickness of the walls being usually from 8 to 12 inches. The walls are bound together by angle irons and tie-rods, or, in some cases, by a complete casing of sheet iron. The latter strengthens the furnace, but radiates more heat than the bricks. The firepot is made any size desired, depending on the size of the establishment, and whether bullion is to be melted in large quantities. The grate bars are separate and removable, their ends resting on narrow iron ledges built into the furnace walls.

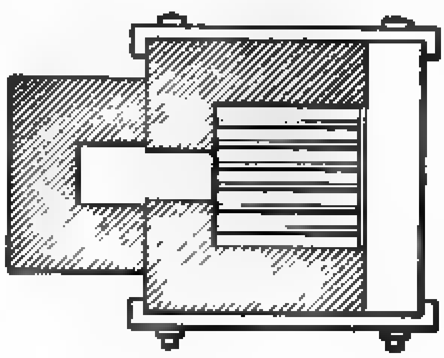


Fig. 19

The draft is regulated by the iron door of the ash-pit. The top of the furnace may be either flat or inclined as shown, with a ledge in front on which to set crucibles and molds. It is made of a cast-iron plate, with the opening usually flush with the sides of the firebox. This opening is covered, while working, with a cast-iron plate or lid. In large furnaces, this plate is sometimes lined with firebrick, and slides back and forth on rollers. The flue connects with the firebox at the back, near the top, and is usually from 16 to 20 square inches in area (4 in. × 4 in., 3 in. × 6 in., or 4 in. × 5 in.) for a 12″ × 12″ furnace.

The stack should be 5 in. × 6 in. or 6 in. × 6 in. in the clear, and should be at least 20 feet high—the higher the better.

For crucible furnaces intended for melting quite large charges, the covers are frequently provided with iron rings, and are hoisted off and removed by means of a small tackle hanging from a trolley that runs on a rail supported over the furnace. The same trolley may be employed for hoisting the pots from the furnace.

FIG. 20

Small portable crucible furnaces for assaying are made of fireclay tile, bound together with a sheet-iron casing or with wrought-iron bands.

When gasoline or gas is used to fire crucible furnaces, a special style of furnace is sometimes employed, in which four to eight crucibles are placed in a small chamber covered with a firebrick lid. The fire enters from the burner at *a*, Fig. 20, passes around the crucibles *b*, and out through a small flue *c*. The crucibles are removed by lifting off the cover shown in dotted lines and handling them with tongs. The shape of the furnace inside permits the two crucibles near the outlet *c* to check the flame and do better work than if the crucible compartment was rectangular. There is a removable shelf *d*, on which the hot crucibles or the cover may be placed for convenience.

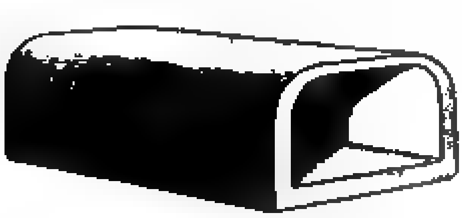


FIG. 21

26. Muffle Furnace.—Muffles are made of fireclay, given the shape shown in Fig. 21, and then burned in a kiln. The front of the muffle opens into the air, and a small hole or slit in the rear permits a constant current of fresh air to flow through the muffle while working. The heat of the

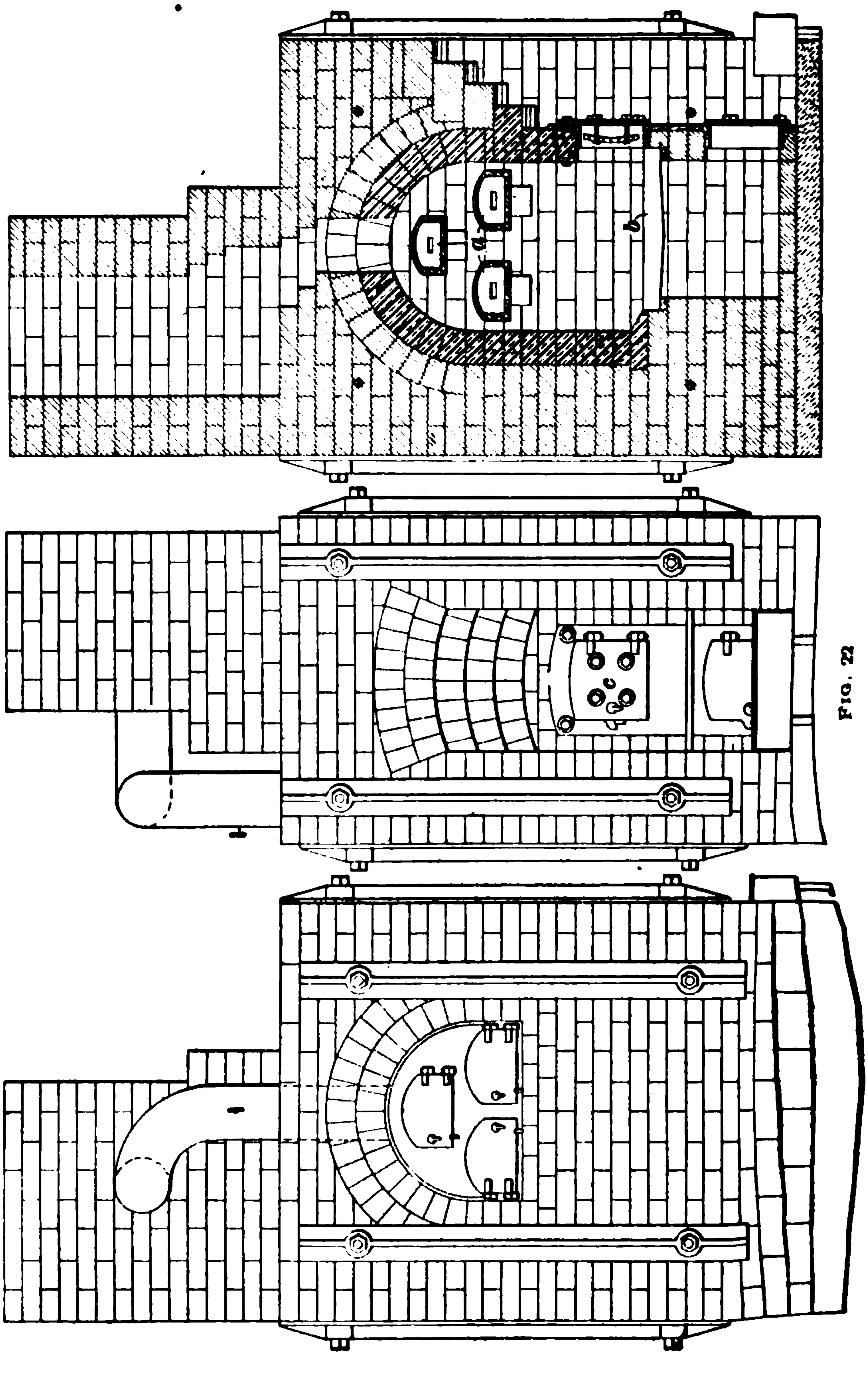


Fig. 22

muffle and the draft through it may be regulated by the door in front. Assay muffles are made in various sizes, from 7 inches long by 3½ inches wide to 18 inches long by 14 inches wide. The 12″ × 6″ and 14″ × 8″ sizes are the most commonly used, as they are plenty large enough to receive a double row of 10-g. crucibles—the size most commonly used in assaying. Muffles usually last only a few weeks, but are easily replaced when they break.

After an ore charge has been smelted in the crucible furnace, the button obtained is placed in a cupel and melted in the muffle furnace, thus necessitating the use of two furnaces. Fig. 22 shows the construction of a brick muffle furnace built for three muffles and to burn bituminous coal. The construction of muffle furnaces varies somewhat with the nature of the fuel used. Furnaces for burning long-flaming, bituminous coal have the muffle *a* 12 to 18 inches above the grate bars *b*, and use a comparatively thin bed of fuel, depending on the flame to heat the muffle. The fire-door *c* is on the level of the grate *b*. In furnaces for burning coke, charcoal, or anthracite, on the other hand, the muffle is set within 6 or 8 inches of the grate, and the fuel is packed in around the muffle. These fuels are all short flaming, and their heat, though intense, is only local, and will scarcely raise a glow in a muffle placed a few inches above the fire-bed—hence the necessity of surrounding the muffle with fuel. The fire-door in furnaces of this type is placed some distance *above* the muffle, and a narrow horizontal slit is made in the furnace on the level of the grate bars for stirring the fire and cleaning the grate. Stationary muffle furnaces are usually built of red brick, lined with one course of firebrick. The walls are firmly braced with buckstays and tie-rods.

The sheet-iron pipe shown in Fig. 22 is for the purpose of drawing off any fumes that tend to escape from the front of the muffle and conducting them to the chimney.

27. In Fig. 23 is shown a two-muffle furnace that has given satisfaction in the West. The muffles are shown at *a*,

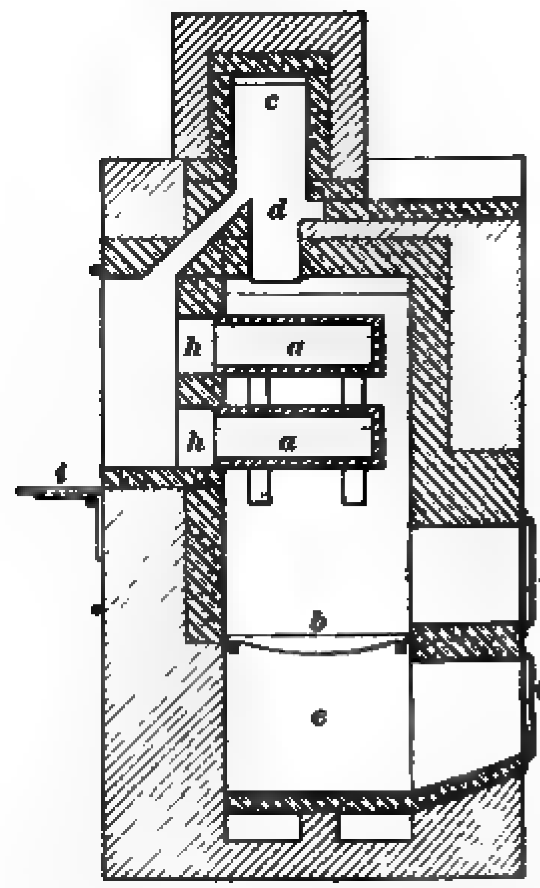


Fig. 23

Fig. 24

the grate bars at *b*, the flue at *c*, the damper at *d*, and the firebox at *e*. It will be noticed that the grate bars are almost on a level with the door *f*, in order that the flames from the soft coal may pass around the muffles and thus obtain the full heat of combustion. The ash door is at *g*, while the muffle

FIG. 25

doors *h* are on the opposite side of the furnace. The muffle furnace shelf *i* is cast iron and is fastened firmly to the furnace wall. The furnace uses from 175 to 250 pounds of coal per day, when making one hundred assays.

28. Portable Muffle Furnaces.—Portable muffle furnaces are made in all sizes, and are designed for burning coke, charcoal, or anthracite. Fig. 24 shows one built of burned fireclay and bound with wrought-iron straps. The muffles *a* of these furnaces are made large enough to receive several crucibles. The fire-door *b* is above the muffle, while the draft holes *c* are below. The fire is stirred by pokers thrust through the draft holes, the ashes being removed through the door *d*.

Where portable muffle furnaces are used, they are made to do double service; that is, smelting and cupeling.

29. Combination Furnaces.—Fig. 25 shows a combination crucible and muffle furnace for coal, charcoal, or coke. It is made of thick sheet iron, lined with firebrick, the lining being about 2½ inches thick. The doors are of wrought iron for the muffle compartment, which is below the crucible compartment. The top of the furnace is fitted with a cast-iron frame with two flat cast-iron covers, lined with asbestos. The covers slide to the right from the center on rails. The crucible part of the furnace is 17 inches square. In the figure, crucibles are seen in the muffle instead of cupels, the inference being that this furnace can smelt in the muffle as well as cupel. The muffle is 15 in. × 9 in. × 5½ in. in dimensions. The price of the furnace described is $60, but a smaller one, having a crucible compartment 12 inches square inside the lining, and taking muffles 6 in. × 12 in. × 4 in., can be had for $30.

30. Gasoline Combination Furnaces.—When the number of assays are few, the gasoline or the oil furnace, constructed with crucible and muffle compartments, finds favor. Such furnaces are ready for use in a short time after firing, as the walls do not require long heating before they assume the proper temperature for assaying. With such a furnace as that illustrated in Fig. 26, both cupeling and melting can be carried on at the same time, thus permitting a number of charges to be melted during the day.

The peculiar construction of the crucible compartment *a*, and of the combustion chamber about the muffle *b*, is such

that the flame is so distributed that the muffle becomes hot all over in a short time after the bottom is heated. The flame from the burner enters the crucible compartment at *c*, and passes around the crucibles, then around the muffle, and finally leaves through the sheet-iron flues shown above and to the rear of the muffle.

Fig. 26

31. Gas and Gasoline Burners.—Assay furnaces that use gas, gasoline, or oil for heat must be provided with proper burners. The gas burner requires a blower *d* to furnish air for the combustion of the gas as it leaves the gas jet. The usual source of power for the blower is electricity; but the blower may be belted to steam power or to a small gas engine. The air enters the burner *e* from the pipe *f*, while the gas enters from the pipe *g*. The proper quantities of air and gas for combustion are regulated by the valves *h* and *i*.

Gasoline burners require that gasoline be vaporized and forced from a small jet orifice by air or steam. For perfect combustion there must be a regulation of air and vapor, and this is accomplished in various ways. The valve for controlling the vapor is delicate, being not much longer than a needle, and is almost as sharp pointed.

Gasoline burners require a tank for the liquid, a pressure gauge and air pump attached to the tank, together with suitable valves and fittings for coupling to the tank. Gasoline or naphthaline for such burners should have a gravity of 72°, Baumé scale.

Fig 27

32. Oil Burners.—When crude petroleum, refined petroleum, or petroleum distillates ranging from 30° to 60° Baumé are used for fuel, special burners and arrangements are required to vaporize the oils. Fig. 27 shows an oil

burner set up ready for use in assaying. Vaporized oil does not produce the quick, intense heat of gasoline; nevertheless, the saving in cost of fuel offsets the slight difference in the time of heating the crucibles; hence, oil burners are to be recommended in places other than those where it is difficult to obtain gasoline at a reasonable price. To use the apparatus, water is placed in the tank *a*, and oil in the tank *b*. A pressure of air is maintained on top of the water and oil

FIG. 28

by the air pump *c*, and the pressure is equalized in the two tanks by the connecting pipe *d*. The air pressure forces the oil through the pipe *e*, and the water through the pipe *f*, to the injector *g*. Both the air and the water are forced into a spray by issuing from the needle valves *h*. This spray is directed by a nozzle through a hole in the furnace front, and when entering this hole entrains sufficient air for combustion if the valves are properly regulated.

An enlarged **crude-oil burner** is shown in Fig. 28. The water enters the tuyère or furnace face *a* through the pipe *b*,

and flows through the pipe *c* to the needle valve *d*. The water is heated in the furnace face by the reflection from the heat in the furnace, and issues from the nozzle *e*, where it is immediately converted into steam. The oil enters the burner through the pipe *f*, and is atomized by the needle valve *g*. The mixture of atomized oil and steam is forced from the nozzle of the burner, and injected through the tuyère hole into the furnace. Sufficient air for combustion is drawn into the furnace by the flame entering. The oil does not burn inside the injector, so that the burner does not clog with soot. Two 10-gallon tanks are loaded with oil and water to a combined height that will furnish about 5 gallons of air space. As the consumption of oil is about twenty times as great as that of water, it is customary to charge the oil tank more heavily than the water tank. When air is pumped to a pressure of 25 pounds to the square inch, it will remain at a pressure high enough to atomize the liquids for about 1 hour, after which more air must be added. There is not much loss of air from good tanks and connections, but the expansion due to the atomizing operation decreases the pressure.

ASSAY DISHES

33. Crucibles.—For fire-assay work, the French clay, the Battersea, or English, the Hessian, or German, and the American, or fireclay, crucibles are used. The foreign crucibles are made usually triangular in shape at the top or with lips for pouring; the American crucibles are made as shown in Fig. 29. The sizes, capacities, number, and weight are given in Table I.

Fig. 29

The size known as the 10-g. crucible will take the ore pulp and fluxes for a ½ A. T.; while the 20-g. crucible is sufficiently large for making a gold-silver assay when 1 A. T. of pulp is used. The crucibles are low, and broad at the base, and for muffle work are safer and more convenient than the narrow-

base form known as the French shape. The narrow-base crucibles are better for furnace work, where the crucible must be thrust into the hot coal, as on sudden heating the broad base of the American-shaped crucible is liable to crack. Care must be taken with either shape, and they should be heated before being thrust into hot coals. Fireclay crucibles are strong, smooth, and pour cleanly.

34. Hessian Crucibles.—Hessian, or German, crucibles are said to be made of three parts of clay and one part of sand. They can be obtained in this country, but on account of their small base, which renders them liable to upset, they

TABLE I

CRUCIBLE DATA

Capacity Grams	Height Inches	Diameter Inches	Quantity in a Barrel	Weight per Barrel Pounds
5	$2\frac{5}{8}$	$2\frac{3}{8}$	900	300
10	3	$2\frac{5}{8}$	630	275
12	$3\frac{1}{4}$	$2\frac{3}{4}$	550	270
15	$3\frac{1}{2}$	$2\frac{7}{8}$	500	280
20	$3\frac{3}{4}$	3	400	290
30	$4\frac{3}{4}$	$3\frac{1}{4}$	280	295
30	$3\frac{7}{8}$	$3\frac{1}{2}$	280	295
40	$5\frac{5}{8}$	$3\frac{3}{8}$	200	255

are not adapted to muffle furnaces. Another objectionable feature is that the crucibles are rough and do not pour cleanly.

35. Graphite, or Plumbago, Crucibles.—Graphite, or plumbago, crucibles are made of from one to seven parts of refractory clay, and from three to ten parts of graphite, with possibly a little sand. If the crucibles contain much silicious matter they are likely to be acted on by the slag of the fusion and be destroyed.

TABLE II

DATA ON GRAPHITE CRUCIBLES

Number	Holding Capacity Liquid Measure			Height Outside	Diameter at the Top Outside	Diameter at the Bilge Outside	Diameter at the Bottom Outside
	Gal.	Qt.	Pt.	Inches	Inches	Inches	Inches
0				2	$1\frac{1}{2}$	$1\frac{5}{8}$	$1\frac{1}{4}$
00				$2\frac{3}{8}$	$1\frac{7}{8}$	$1\frac{7}{8}$	$1\frac{3}{8}$
000				$2\frac{1}{2}$	$1\frac{7}{8}$	$2\frac{1}{8}$	$1\frac{1}{2}$
0000				3	$2\frac{3}{8}$	$2\frac{1}{2}$	$1\frac{3}{4}$
1				$3\frac{5}{8}$	$3\frac{1}{8}$	3	$2\frac{1}{4}$
2				$4\frac{1}{2}$	$3\frac{3}{4}$	$3\frac{5}{8}$	$2\frac{3}{4}$
3				$5\frac{1}{4}$	$4\frac{1}{4}$	$4\frac{1}{8}$	3
4				$5\frac{5}{8}$	$4\frac{5}{8}$	$4\frac{1}{2}$	$3\frac{1}{4}$
5			$1\frac{1}{2}$	6	$4\frac{7}{8}$	$4\frac{3}{4}$	$3\frac{1}{2}$
6		1		$6\frac{1}{2}$	$5\frac{1}{4}$	$5\frac{1}{8}$	$3\frac{3}{4}$
7		1	$\frac{1}{4}$	$6\frac{3}{4}$	$5\frac{1}{2}$	$5\frac{1}{2}$	4
8		1	$\frac{1}{2}$	$7\frac{1}{4}$	$5\frac{3}{4}$	$5\frac{3}{4}$	$4\frac{1}{4}$
9		1	$\frac{3}{4}$	$7\frac{5}{8}$	6	$6\frac{1}{4}$	$4\frac{1}{2}$
10		1	1	8	6	$6\frac{1}{2}$	$4\frac{3}{4}$
12		2		8	$6\frac{1}{4}$	$6\frac{3}{4}$	5
14		2	1	$8\frac{1}{2}$	$6\frac{3}{4}$	$7\frac{1}{4}$	$5\frac{1}{2}$
16		2	1	$8\frac{3}{4}$	7	$7\frac{1}{2}$	$5\frac{1}{2}$
18		3	1	$9\frac{3}{4}$	$7\frac{3}{8}$	8	$5\frac{3}{4}$
20	1			$10\frac{1}{4}$	$7\frac{3}{4}$	$8\frac{3}{8}$	6
25	1		1	$10\frac{1}{4}$	8	$8\frac{5}{8}$	$6\frac{1}{4}$
30	1	1	1	11	$8\frac{5}{8}$	$9\frac{1}{4}$	$6\frac{1}{2}$
35	1	2	1	$11\frac{5}{8}$	$9\frac{1}{4}$	$9\frac{3}{4}$	7
40	2			$12\frac{3}{8}$	$9\frac{1}{4}$	$10\frac{1}{4}$	$7\frac{3}{4}$
45	2	1		13	$9\frac{3}{4}$	$10\frac{1}{2}$	$7\frac{5}{8}$
50	2	3		$13\frac{1}{2}$	$10\frac{1}{8}$	$11\frac{1}{4}$	$7\frac{7}{8}$
60	3			14	$10\frac{3}{8}$	$11\frac{5}{8}$	8
70	3	1		$14\frac{1}{2}$	$10\frac{7}{8}$	12	$8\frac{1}{2}$
80	3	2	1	$15\frac{3}{8}$	$11\frac{1}{8}$	$12\frac{3}{8}$	$8\frac{5}{8}$
90	4			$15\frac{7}{8}$	$11\frac{1}{2}$	$12\frac{1}{2}$	9
100	4	2	1	16	$11\frac{7}{8}$	$13\frac{1}{8}$	$9\frac{3}{8}$
125	4	3	1	$16\frac{5}{8}$	$12\frac{1}{2}$	$13\frac{3}{4}$	$9\frac{1}{2}$
150	6	3		$18\frac{1}{4}$	$13\frac{1}{4}$	$14\frac{3}{4}$	$10\frac{3}{8}$
175	7	3	1	$19\frac{1}{2}$	$14\frac{1}{4}$	$15\frac{3}{4}$	$10\frac{3}{4}$
200	9	3	1	$20\frac{1}{2}$	15	$16\frac{1}{2}$	$11\frac{1}{4}$
225	10	1	1	$20\frac{3}{4}$	$15\frac{1}{4}$	$16\frac{3}{4}$	$12\frac{1}{2}$
250	10	3		$20\frac{1}{2}$	$15\frac{1}{4}$	17	$11\frac{3}{4}$
275	11	3		$22\frac{3}{8}$	15	$16\frac{5}{8}$	$12\frac{1}{2}$
300	12	2		22	$16\frac{1}{4}$	$17\frac{1}{2}$	$12\frac{1}{2}$

Graphite crucibles are not used for assaying, but for melting bullion, an operation that frequently the assayer is called on to perform. In Table II are given the sizes of graphite crucibles.

36. Platinum Crucibles.—Platinum crucibles are sometimes used in gravimetric assays by fusing in them ores, slags, and furnace products with carbonate of soda. Platinum crucibles are sold by the gram, and are expensive although durable dishes. They weigh with their covers as many grams as they hold cubic centimeters. The crucibles may be cleaned by heating them in a little nitric acid and scouring them afterwards with iron oxide. Platinum crucibles should never be handled roughly, and should be cleaned as soon as possible after use. Lead, zinc, tin, nickel, copper, and silver will alloy with platinum, and should not be fused in platinum crucibles. Caustic soda, caustic potash, or potassium cyanide should not be fused in platinum crucibles, as they make the platinum brittle.

37. Silver Crucibles.—Silver crucibles are sometimes used where caustic soda and caustic potash are the flux since these fluxes slag the silica in porcelain crucibles. The crucibles with a gold lining are preferable and the heat should be raised gradually to prevent the silver melting. To ascertain the percentage of gold extracted by a cyanide solution, dishes made of silver foil are used. These are called *boats*. Alcohol lamps or sand baths are to be used for heating silver crucibles; for any sulphur products from gas or oil will corrode them, and nearly all gas and petroleum contain sulphur in small quantities, which forms a sulphur gas when ignited.

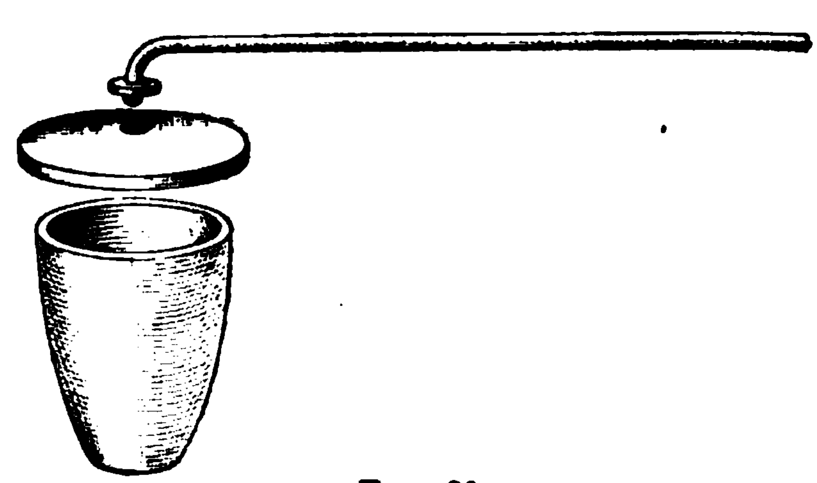


FIG. 30

38. Porcelain Crucibles.—Porcelain crucibles are used for parting the gold-silver beads after inquartation, and in annealing the gold obtained from an assay after

inquartation and before the final weighing. They are also used for the ignition of precipitates that cannot be made in a platinum or silver crucible. A porcelain crucible, with a perforated cover and stem for the escape of gas, is shown in Fig. 30. Such crucibles, termed **Rose crucibles,** are used when it is desired to ignite precipitates in an atmosphere of hydrogen or sulphureted hydrogen. A substitute for Rose's crucible is a porcelain crucible with an ordinary clay pipe for cover. Silica should not be fused with caustic soda or potash in porcelain crucibles.

FIG. 31

39. Scorifiers.—The usual form of a scorifier is shown in Fig. 31. These are made of good fireclay, thoroughly mixed and burned, and come in the sizes, weights, and prices quoted in Table III.

TABLE III

DATA ON SCORIFIERS

Outside Diameter Inches	Number in the Barrel	Weight per Barrel Pounds	Outside Diameter Inches	Number in the Barrel	Weight per Barrel Pounds
2	2,400	335	3	1,000	300
$2\frac{1}{4}$	2,000	287	$3\frac{1}{2}$	800	300
$2\frac{1}{2}$	1,900	317	4	600	300
$2\frac{3}{4}$	1,400	303			

Scorifiers are used to make assays of rich ore. Those most commonly used are from $2\frac{1}{4}$ to $2\frac{3}{4}$ inches in outside diameter. Such sizes will take $\frac{1}{10}$ A. T. with the necessary fluxes. For the purpose of putting the scorifiers in the muffle, the special tongs shown in Fig. 35 are needed.

40. Roasting Dishes.—Roasting dishes are saucers molded from fireclay and kiln burned. They range in

size from 2 to 7 inches in outside diameter. While not indispensable, they are very convenient for roasting sulphide ores, or for drying or calcining ores or other materials.

41. Cupels.—**Cupellation** is the process of oxidizing base metals, and separating them from the non-oxidizable metals gold and silver. Cupels made of bone ash possess a porosity that permits the oxidized base metals to sink into them, while the gold and silver remain in the cupel bowls. Fig. 32 shows the most common form of the bone-ash cupels used in separating the gold and silver from the lead buttons. These cupels may be purchased ready made, but most assayers prefer to buy the bone ash in bulk and make their own cupels. The home-made cupels are much cheaper, and if well made and dried, are fully as serviceable as the purchased articles. All the tools required are a brass or iron mold and pestle (shown in Fig. 33) and a wooden mallet.

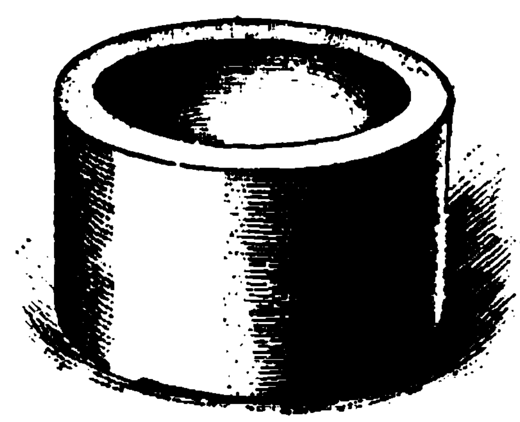


FIG. 32

A pound or so of bone ash is thoroughly mixed with just enough water to dampen it, so that when squeezed in the hand it will stick together and show distinctly the impression of the fingers. It must not, however, contain enough water to feel wet and dampen the fingers; the proper consistency is difficult to describe, but it is soon learned. The cupels will be stronger and less liable to crack in drying if a strong solution of carbonate of soda or potassium (sal soda or pearlash) is used for moistening the bone ash instead of water alone. The moistened bone ash is sifted through an ordinary flour sieve to break up all lumps. To make the cupel, the cupel ring is placed on a smooth block of wood and filled level full with the moistened bone ash put in loosely; the pestle or plunger is then inserted and struck four or five moderate blows with the mallet,

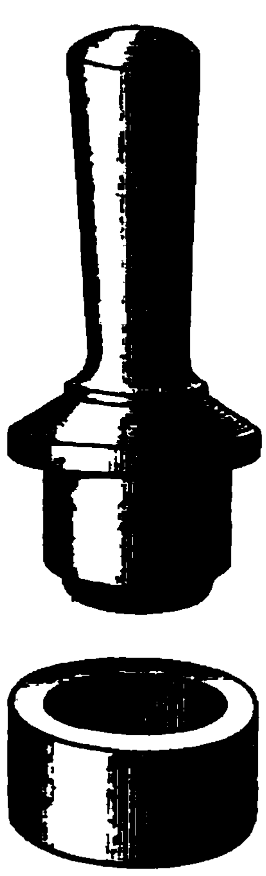


FIG. 33

compressing the ash into the form of the cupel. To remove the cupel from the mold, the mold is inverted; then with a gentle upward pressure the plunger is turned so as to free it from the cupel, and the cupel is gently but steadily forced out.

The moist cupels are set aside and allowed to dry out slowly. If it is absolutely necessary to use them at once, they may be dried out and made reasonably safe by setting them on top of the furnace for several hours while running; but, if possible, they should be allowed to stand several weeks before using. The British mint keeps the cupels 2 years before using them.

The texture of the cupel is very important. If it is too porous, it will absorb some silver, and the results of the assay will be too low; if too dense, it will crack or "check" in the muffle when it becomes saturated with litharge. The density depends on the fineness of the bone ash, the amount of water used in mixing, and the amount of compression. The finer the bone ash, the more dense the cupel; and the damper the ash, the greater will be the compression from the same power on the plunger. The cupels should be made of a medium grade of bone ash, or the mold may be filled about two-thirds full with coarse ash and the remainder with fine. A cupel made in the latter way will absorb a great deal of litharge and very little silver. If about one part of common flour to ten of bone ash is thoroughly mixed with the bone ash *before moistening*, the cupels may be compressed rather harder than when bone ash is used alone; then, when placed in the muffle, the flour will burn out, leaving the cupel quite porous and in good condition for absorbing litharge.

The shape of cupels is important. That shown in Fig. 32 is the most common, because it is readily removed from the mold and is easily handled in the muffle without risk of tipping. The mold is sometimes made so that the cupel is a trifle wider at the base than at the top; this makes it easier to remove from the mold, but the cupel must then be more carefully handled in the muffle, as the tongs grip the sloping sides of the cupel along their lower edge, and if the point of contact happens to come below the center of gravity of the

cupel, the latter will be likely to turn upside down and lose the button. The bowl of the cupel should be large enough to hold the melted lead button without overflowing, and the cupel should weigh when dry approximately at least as much as the lead button to be cupeled in it. The usual diameter is $1\frac{1}{4}$ inches. If a button is too large to be absorbed by one cupel, it may be cupeled in two portions; or, if the bowl of the cupel is large enough to hold the lead, the button may be put in one cupel, and this set on a second, the latter being upside down in the muffle. As soon as the upper cupel is saturated with litharge, the excess will be absorbed by the under cupel.

42. Porcelain Capsules.—Small porcelain crucibles, or capsules, are used in parting the gold and silver. The parting may be done, if desired, in test tubes or small parting flasks, and the gold afterwards annealed in small, porous, clay *annealing cups*. The use of porcelain capsules is preferable, however, as they can also be used for annealing; the parted gold is washed in the capsule by filling it with water and decanting (pouring off) the washings, repeating several times; the gold is then annealed in the muffle or over the blast lamp without removing it from the capsule. The porcelain capsules are, moreover, much stronger and more durable than the annealing cups, and no more expensive.

FURNACE TOOLS

43. Firing Tools.—Firing tools—poker, shovel, and scraper—are, of course, necessary in connection with assay furnaces using solid fuel.

44. Crucible Tongs.—When a crucible furnace is used, a pair of crucible tongs is necessary for lifting the crucibles. These are of various designs. Fig. 34 (*a*) shows the double-bent and (*b*) the single-bent crucible tongs commonly used for handling medium-sized melting crucibles. These tongs are made of wrought iron and are from 30 to 36 inches long.

A pair of small crucible tongs, Fig. 34 (*c*), is used for handling small porcelain crucibles, for removing nails from assay crucibles, and for handling annealing cups. These

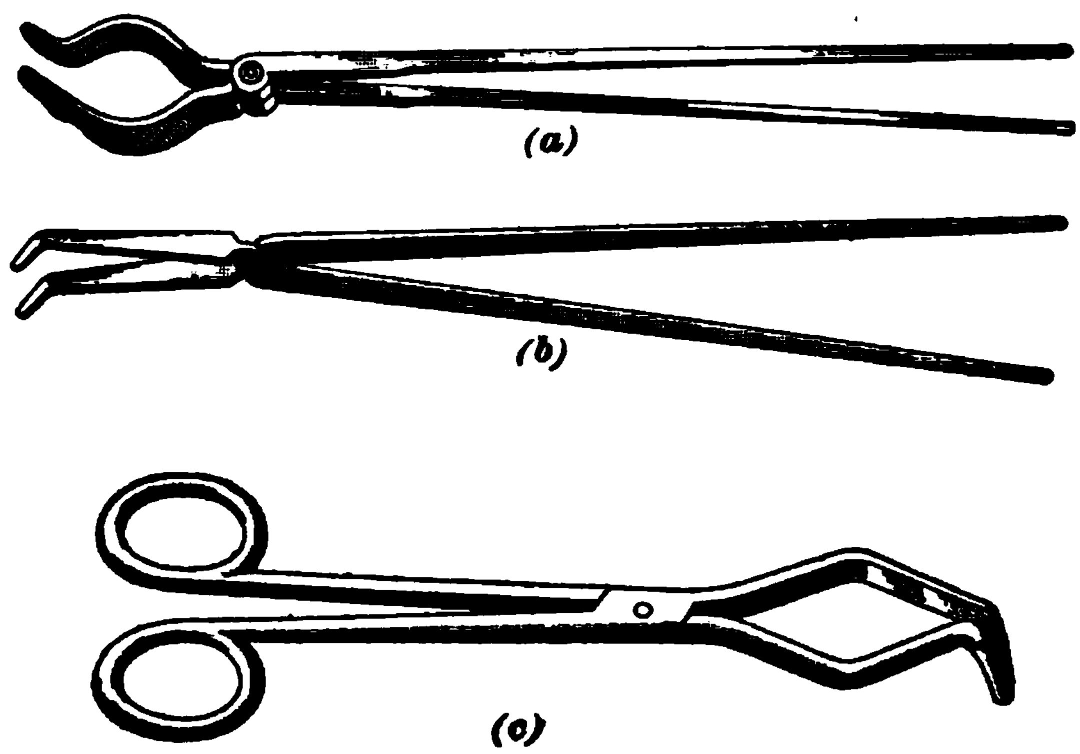


Fig. 34

tongs are about 8 inches long, and are platinum-tipped, which makes them expensive, their price being approximately $5.

45. **Scorifier Tongs.**—Fig. 35 (*a*) shows the common form of tongs used for handling the scorifiers and the small crucibles used in muffle work. These tongs are made of steel, or of wrought iron with a steel spring, and are from

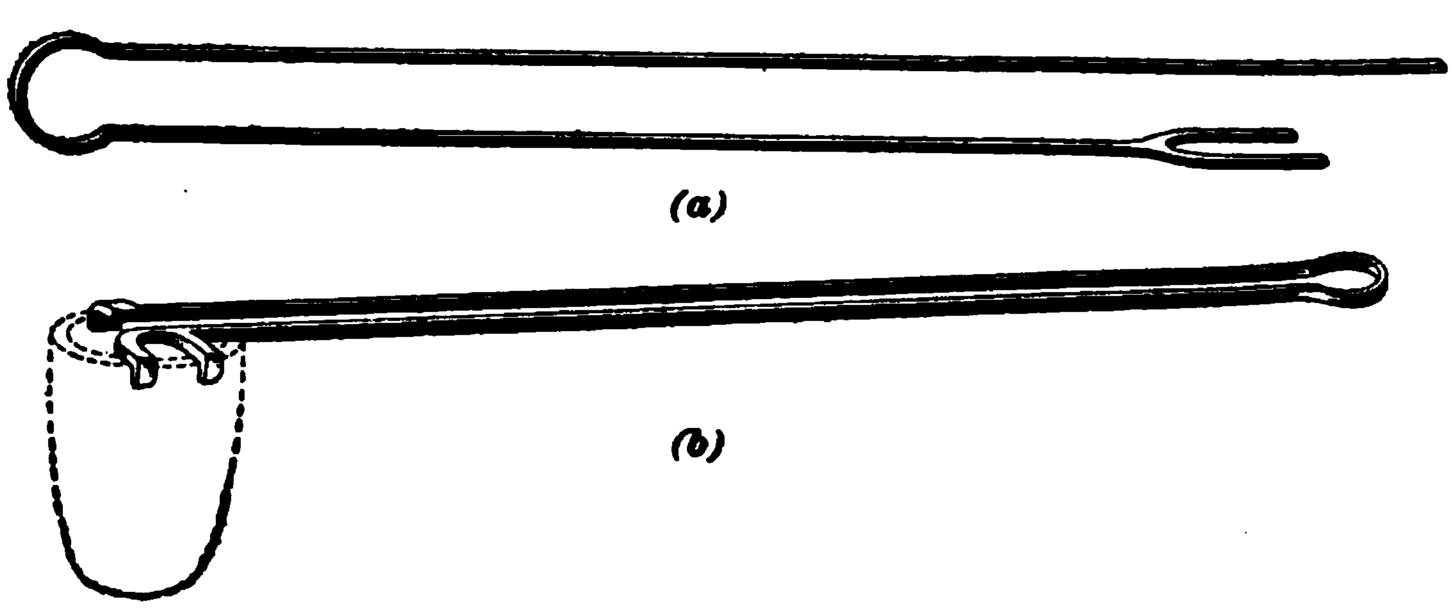


Fig. 35

27 to 36 inches in length. They are large enough to handle 20-g. crucibles, the largest size used in muffle work, and are much handier for that purpose than crucible tongs. Fig. 35 (*b*) shows a pair of scorifier tongs that grasp the dish on the

upper rim. They are not as sure nor as readily applied as the tongs in Fig. 35 (*a*), and cannot be used for crucibles with safety, although shown attached to the outline of a crucible.

46. Cupel Tongs.—Fig. 36 shows the ordinary form of cupel tongs for handling the bone-ash cupels. Like the

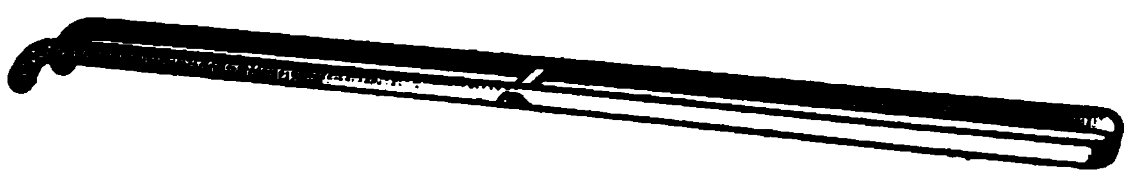


FIG. 36

scorifier tongs, they are made entirely of steel. A steel guide, fixed in the middle of one leg and passing through a hole in the other, serves to keep them in line. Scorifier tongs are also frequently fitted with similar guides. Cupel tongs are made, as shown, with a bend at the end, so that cupels at the back of the muffle can be removed without disturbing those in front.

47. Molds.—Molds made of cast iron are used to receive the melted contents or scorifiers and crucibles. Their use hastens the cooling of the charge and leaves the crucibles in such a condition—when the fusion has been

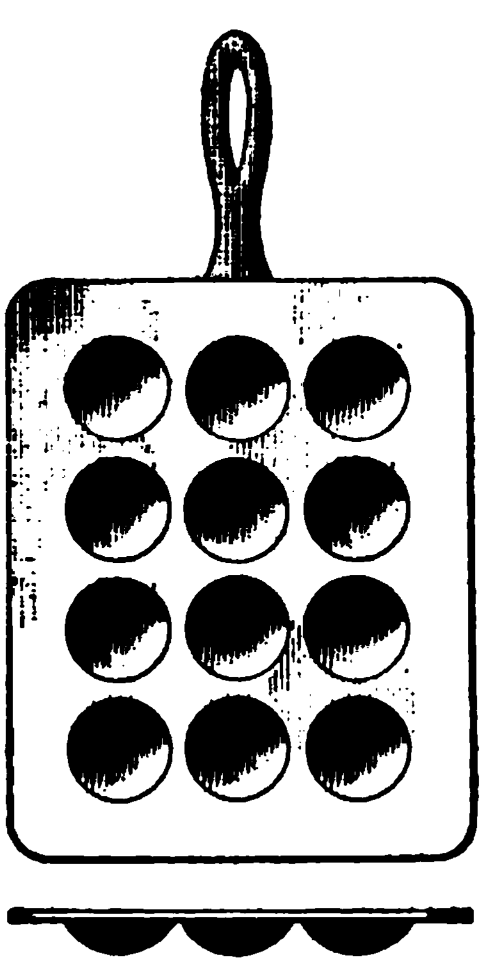


FIG. 37

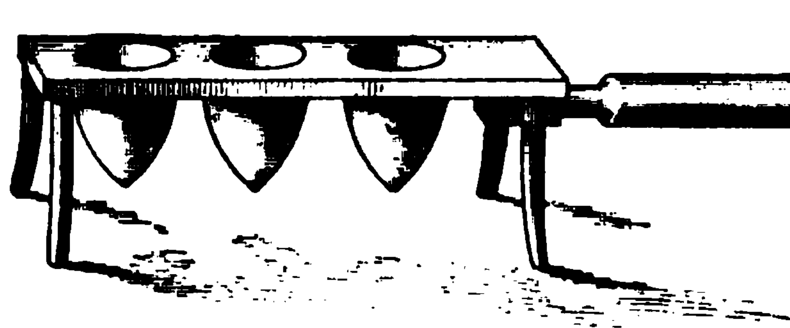


FIG. 38

perfect and the pour clean—that they can be used again if desired. Scorifiers, however, should never be used for more than one fusion, as the scorification usually corrodes the vessel so much as to render it unsafe for use a second time.

In custom assay offices, new crucibles should be used for each charge, and only in mine assay offices, where the same kind of ore is assayed, can old crucibles be used. Umpire assays and check-assays cannot safely be run in old crucibles, no matter how clean the pour may be.

Fig. 37 shows a 12-hole mold for ordinary work with scorifiers and 10-g. crucibles. For larger crucibles, a mold with deeper holes must be used. Fig. 38 shows one form of crucible mold.

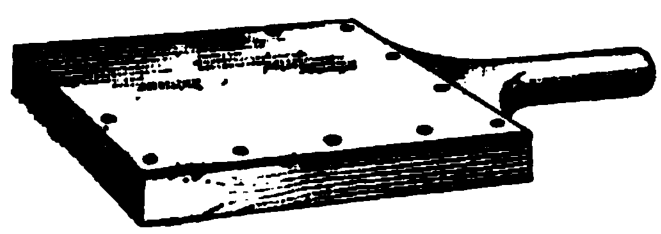


FIG. 39

48. Cupel Board.—An almost indispensable part of the assayer's outfit is the cupel board, or hot board, on which hot cupels are set as they are withdrawn from the furnace. It is merely a piece of 1-inch board, about 10 inches wide, with a handle at one end, as shown in Fig. 39, and with a rectangular piece of $\frac{1}{8}$-inch sheet iron, about 10 in. × 12 in., screwed to the upper side. It is a good plan to cut a number of small, radiating grooves in the board under the plate, to allow the escape of the gas that forms at first, before the wood has become charred, when the iron plate becomes heated from the red-hot cupels. The gas

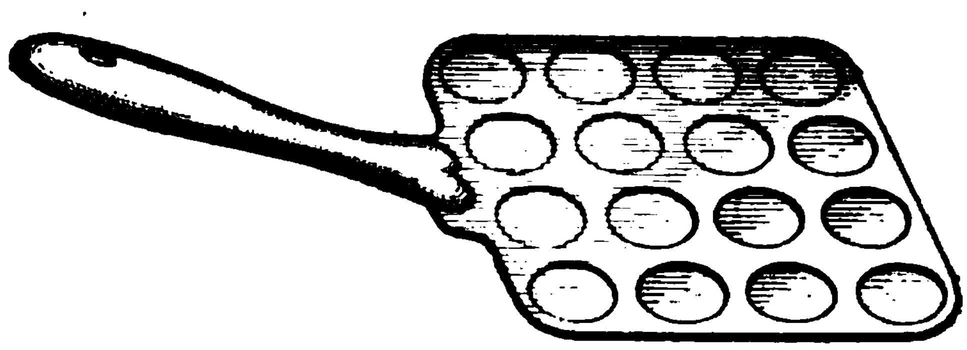


FIG. 40

will usually escape without this precaution, but occasionally it accumulates between the board and the plate and explodes. The explosion is not violent enough to injure the board, but it will upset and mix the cupels, making it necessary to repeat a number of assays. Fig. 40 shows a cast-iron tray for cupels. Cupels are numbered and placed on the trays according to their numerical order. The trays are sometimes marked with chalk to indicate the position that the cupel is to occupy.

49. Hammer.—A hammer is necessary for beating out lead buttons, to free them from slag and get them into convenient shape for cupellation. A 2-pound machinist's hammer with a square face is of a convenient size and shape for this purpose.

50. Button Tongs.—A pair of spring button tongs, Fig. 41, is necessary for handling the button while beating it out.

FIG. 41

To remove the gold-silver bead from the cupel, a pair of pliers is used, as shown in Fig. 42. These will crack any slag or bone ash adhering to the bead and permit it to be brushed off by the bead brush shown in Fig. 43, before weighing.

Fig. 44 shows one side of an assay laboratory that is an adjunct to a chemical laboratory. The floor is cement and the side walls are covered with sheet metal wherever there is danger from fire. The pulp weighing room is to the left of the partition against which the shelves carrying dishes is shown. The operator shows the proper position for holding the muller when bucking samples. To the right of the bucking

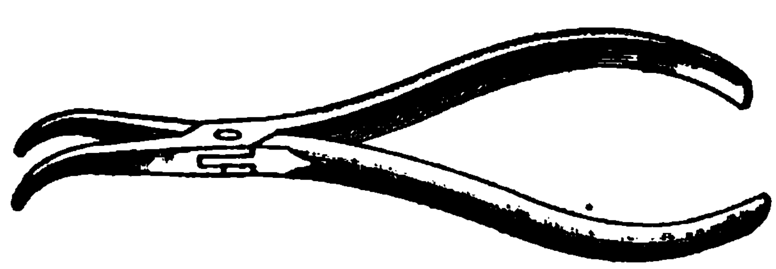


FIG. 42

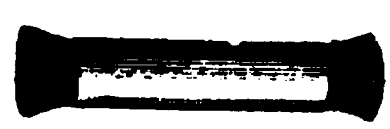


FIG. 43

board is a pan for catching the rock coming from the hand-power crusher. As each sample is crushed it is turned on the bucking board, but previous to bucking the larger pieces of rock are reduced by the hand or button hammer shown leaning against the wall. This method of breaking samples greatly reduces the labor of bucking. Cupels, scorifiers, crucibles, and roasting dishes are kept in sight on the shelves, while the extra furnace muffles are kept on top of the cupboard; if, therefore, the assayer runs out of any such articles he is to blame. Back of the operator and against the wall the sieve used for sizing may partly be seen. On the left-hand front corner of

Face 44

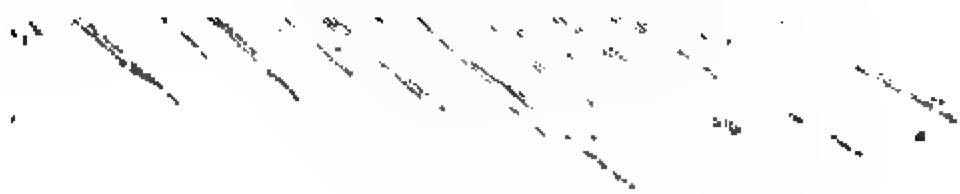


FIG. 4A

the table is a flat anvil on which the slag is broken free from the lead button and the button also shaped. Boxes for holding slag and rubbish are shown beneath the table. All pulp samples are kept in order in the pulp room, as well as the fluxes.

Fig. 45 shows the side of the room opposite the shelves. The furnace is heated by gas mixed with air. The blower is not shown distinctly, but it is revolved by the motor back of the operator. The furnace, which is of the muffle type, works very satisfactorily. The operator is seen taking a crucible from the muffle for pouring into the button molds on the bench. These molds rest on a slab of stone or cast iron to prevent the heat from the melt setting fire to the bench. The button tray is seen hanging on the window box. The door to the furnace is on the bench back of the molds. The cylinder under the furnace is the chamber in which air and gas are mixed previous to being burned in the furnace. The hood above the furnace is for the purpose of drawing the hot air from the room up the chimney. Fresh air is brought from outside the building to supply the blower, as that air is not so rarified and can supply more oxygen for combustion per cubic foot. The dimensions of the furnace are such that it occupies a floor space of 5.638 square feet. It has a height of 57 inches, and weighs in all about 900 pounds.

WEIGHTS AND MEASURES

ENGLISH AND METRIC SYSTEMS

AVOIRDUPOIS WEIGHT

16 drams (*dr.*)	=	1 ounce	*oz.*	=	28.3495 g.
16 ounces	=	1 pound	*lb.*	=	453.5920 g.
100 pounds	=	1 hundredweight .	*cwt.*	=	45.359 Kg.
20 cwt., or 2,000 lb. . .	=	1 ton	*T.*	=	907.184 Kg.

TROY WEIGHT

24 grains (*gr.*)	=	1 pennyweight . .	*pwt.*	=	1.5552 g.
20 pennyweights	=	1 ounce	*oz.*	=	31.1035 g.
12 ounces	=	1 pound	*lb.*	=	373.2419 g.

MEASURES OF LENGTH (METRIC)

The **meter** is the *unit of length*, and is equal to 39.37 inches, nearly.

10 millimeters (*mm.*) . . .	=	1 centimeter . . .	*cm.*	=	.3937 in.
10 centimeters	=	1 decimeter . . .	*dm.*	=	3.937 in.
10 decimeters	=	1 meter	*m.*	=	3.28 ft.
10 meters	=	1 dekameter . . .	*Dm.*	=	32.8 ft.
10 dekameters	=	1 hektometer . .	*Hm.*	=	328.09 ft.
10 hektometers	=	1 kilometer . . .	*Km.*	=	.62137 mi.
10 kilometers	=	1 myriameter . .	*Mm.*	=	6.2137 mi.

MEASURES OF WEIGHT (METRIC)

The **gram** is the *unit of weight*, and is equal to 15.432 grains, or the weight of a cube of pure distilled water at 4° C., the edge of which is *one one-hundredth* ($\frac{1}{100}$) of a meter.

10 milligrams (*mg.*)	=	1 centigram	*cg.* =	.15 gr.
10 centigrams	=	1 decigram	*dg.* =	1.54 gr.
10 decigrams	=	1 gram	*g.* =	15.432 gr.
10 grams	=	1 dekagram	*Dg.* =	154.32 gr.
10 dekagrams	=	1 hektogram	*Hg.* =	3.53 oz., avoir.
10 hektograms	=	1 kilogram or kilo	*Kg.* or *K.* =	2.20 lb., avoir.
10 kilograms	=	1 myriagram	*Mg.* =	22.05 lb., avoir.

CUBIC MEASURE (METRIC)

1,000 cubic centimeters (c. c. or *cm.*^{3}) = 1 cubic decimeter, or liter (*l.*).
1 liter of water at 4° C. weighs 2.2 lb., avoirdupois.
1,000 cubic decimeters = 1 cubic meter (*cu. m.*), or kiloliter (*Kl.*).
1 kiloliter of water at 4° C. weighs 22.04 cwt.

ASSAY-TON WEIGHTS

Multiples . . . . 4 assay tons = 116.66666 grams
2 assay tons = 58.33333 grams

Unit . . . . . . The assay ton (*A. T.*) is equal to 29.16666 grams.

Subdivisions . .
$\frac{1}{3}$ assay ton = 9.7222 grams
$\frac{1}{6}$ assay ton = 4.8611 grams
$\frac{1}{10}$ assay ton = 2.9166 grams
$\frac{1}{20}$ assay ton = 1.4583 grams

ASSAYING

(PART 2)

ASSAY CHARGES

FLUXING ORES

1. Ore Deposits.—Ore deposits occur as veins in rock fissures, as pockets where the ore has replaced the rock, and as beds. Ore is not pure, but is usually found associated with gangue and other minerals, which are objectionable owing to the fact that they make the reduction of the ore difficult and expensive. In addition to gangue rock, ore—especially ore from narrow veins and from pockets—is frequently found mixed with the country rock.

2. Gangue Minerals.—In the ores of gold, silver, copper, lead, etc., the proportion of the metal minerals to the minerals composing the ore deposit is very small. The gangue is said to be **acid** if quartz predominates, and **basic** if a metallic oxide predominates. If the gangue is acid, basic material must be added to a smelting charge to make it fusible, for quartz alone is not fusible at even the highest furnace temperatures. If, on the other hand, the gangue is basic, acid materials must be added to form a fusible slag.

The gangue as a whole is acid or basic according as the acid or basic constituents are in excess. When both are present in the same proportion, the gangue is said to be *neutral*. All *silicates* are more or less self-fluxing, as the silica is partly neutralized by combination with the metallic

base of the silicate. They may be considered as made up of metallic oxides and silica, with the latter usually in excess; and the metallic oxides may be credited to the bases, as if uncombined, leaving only silica as the acid constituent of gangues. With some few exceptions, the gangue rocks that contain the most valuable metalliferous-ore deposits are silicious in character.

In Table I are given the principal gangue minerals arranged according to their chemical composition and action.

TABLE I

PRINCIPAL GANGUE MINERALS

Acid	Basic
Silica, uncombined; as, quartz crystals, rock quartz, quartzite, sandstone, etc. Silicates, or silica, combined with a base; as, feldspars, mica, clay, slate, etc. Rocks in which silica, either free or combined, predominates; as, granite, quartz-porphyry, etc. Generally speaking, the acidity of a gangue is due to silica.	Metallic oxides and carbonates, notably those of iron, calcium (lime), magnesium, and manganese. Fluorite, or fluor spar (calcium fluoride). Barite, or heavy spar (barium sulphate). Generally speaking, all the metallic elements and their common salts, with the exception of the silicates, act as bases.

3. Metal-Bearing Rocks.—Nearly all rock formations contain metallic minerals in small quantities, but only certain igneous rocks contain them in their original composition. The decomposition of minerals and their solution by circulating waters, followed by their precipitation from these solutions, resulted in the filling up, or, as geologists say, "healing," of rock fissures, and in the formation of pockets and some bedded-ore deposits.

In Table II, the average composition is given of the leading copper, gold, lead, and silver bearing rocks. Each rock named in this table will vary in the percentage composition of the oxides from which it is formed. The percentages of the constituents of each rock given, represent the average composition as determined by the analysis of several specimens.

TABLE II

COMPOSITION OF ROCKS

Name of Rock	Per Cent. of SiO_2 (Silica)	Per Cent. of Al_2O_3 (Alumina)	Per Cent. of Fe_2O_3 (Iron Oxide)	Per Cent. of $CaO.MgO$ (Alkaline Earths)	Per Cent. of Na_2OK_2O (Alkalies)
Granite . . .	68	15	4	4	8
Syenite . . .	59	17	7	9	7
Quartz diorite	63	17	6	8	6
Diorite . . .	55	18	9	12	5
Gabbro . . .	49	18	9	19	4
Pyroxenite . .	43	4	11	37	1
Quartz porphyry or rhyolite . .	75	13	2	1	8
Trachyte . .	63	17	5	4	10
Dacite	68	15	4	5	7
Andesite . . .	61	16	5	7	6
Basalt	50	14	14	17	4
Serpentine . .	43.48			43.48 MgO	
Dolomite . .				54.35 CaO 45.65 MgO	
Clay	23				
Quartzite . .	77	13	1.1	1.7	5

4. Complex Ores.—Very few metals are found native, but are found as oxides, carbonates, sulphides, etc., for which reason the combinations are termed metallic minerals. It is unusual for metallic minerals to be found pure, in fact they are associated with other metallic minerals, which are in some cases predominant, and in other cases subordinate, in quantity. To assay minerals of this description requires

more care than would be necessary, if there was but one metallic mineral to be determined.

When an ore has a highly silicious gangue, it is mixed with a basic substance, such as lime or sodium carbonate, that forms a silicious slag. If the gangue of the ore is basic, then some acid material, such as silica, must be added to form a fluid slag. Other materials in the ore must also be slagged, in order that the lead used may collect the metal being assayed in a lead button at the bottom of the crucible or scorifier. The necessity of studying the composition of an ore is now evident, for it is from this knowledge that the proper proportions of materials to furnish a fluid slag are determined.

FLUXES

5. Infusible Ores.—The majority of ores are in themselves infusible, or nearly so, at the temperatures obtainable in an assay furnace. If, however, the pulverized ore is well mixed with the correct proportions of certain solid chemical reagents, the mixture will readily fuse at a moderate heat to a fluid mass, called **slag.** Owing to their greater specific gravity, such heavy metals as lead, gold, and silver, which are reduced to their metallic state during the fusion, settle through the slag. The reagents used for this purpose are called **fluxes,** from their property of making the mixture fluid.

Fluxes may, like the ore, be infusible alone, but fusible when mixed in the proper proportions with ore. For example, iron oxide, calcium oxide (lime), and silica (the most common gangue material of ores) are each, separately, extremely infusible, but when properly mixed they form a very fusible slag. These three substances are, as a matter of fact, the principal constituents of all slags except the slag that comes from iron blast furnaces, where the iron is removed by being reduced to a metal at an extremely high temperature and in a very powerful reducing atmosphere, and the slag is made up mainly of silica and lime. As slags are more or less definite chemical compounds, the composition of an ore with regard to lime, iron, and silica is taken

into consideration when computing the furnace charge, and the quantity of fluxes added is just sufficient to give the correct proportions of these substances to produce a slag of fixed composition.

6. General Fluxes.—In employing fluxing materials that are in themselves refractory, an excess of any of them over the correct fixed proportions will make the mixture somewhat less fusible. If, on the other hand, the fluxing material is itself quite fusible, any excess will make the slag more fluid by simple dilution, even if it does not combine chemically with the rest of the slag-making material. In assaying, it is impracticable to determine the composition of each sample before assaying it; hence, the fluxes used are mostly those which are of themselves readily fusible, and mixtures are made as nearly as possible universal; that is, the assayer seeks to find a mixture that will flux all ores. A *perfect* universal flux is out of the question, as there is too wide a variation in the chemical characteristics of ores for any one mixture to flux them all satisfactorily.

Occasionally, when using a general flux, an ore will be met that will not flux satisfactorily. In such a case, the cause of refractoriness is determined either by the eye or, if necessary, by chemical analysis, and is corrected by the addition of the proper fluxing materials. For example, an excess of silica, which is acid in its reactions, may be corrected by the addition of a flux that reacts as a base; on the other hand, any excess of the metallic oxides, which are basic in their reactions, must be counteracted by the use of an acid flux. Fluxes are a mixture of substances, the nature of which is such that a molten slag will be formed so fluid that any metals in the pulp can settle by their greater specific gravity to the bottom of the crucible.

7. Heat of Formation.—When two substances, one of which is acid and the other basic, are fused together, they require less heat than when either is fused separately. Such substances form slags called **silicates**, which have a definite chemical composition. As the proportions in which the two

substances are mixed have a direct influence on the temperature at which they will unite, the heat of slag formation will be increased if one of the substances exceeds the proportion necessary to be maintained in order that they may unite at the lowest temperature. The **eutectic**, or lowest melting temperature, of a combination of fluxes is the heat of formation. In most cases, a mixture of several fluxes will furnish a eutectic below that of two fluxes—a fact that is frequently taken advantage of by the assayer. The heat of formation is sufficient to convert metallic oxides into metals provided a reducer is present; but the mixture of fluxes must be such as to produce a fluid slag.

8. Sodium Carbonate, Na_2CO_3.—When an equal number of molecules of sodium carbonate, or soda, and silica are fused together, a sodium bisilicate is formed, according to the equation:

$$\underset{\substack{\text{sodium}\\\text{carbonate}\\105.31}}{Na_2CO_3} + \underset{\substack{\text{silica}\\59.96}}{SiO_2} = \underset{\substack{\text{sodium}\\\text{silicate}\\121.60}}{Na_2SiO_3} + \underset{\substack{\text{carbon}\\\text{dioxide}\\43.67}}{CO_2}$$

When fusion takes place, the carbon dioxide escapes as gas and causes a brisk effervescence; at the same time the silica fuses with the soda and forms sodium silicate. Sodium carbonate absorbs so much moisture from the atmosphere that it is disagreeable to handle, as it covers the hands and apparatus with a slimy solution of the carbonate. When left in the air, it cakes into solid lumps, and must therefore be kept in air-tight vessels. For assaying purposes it should be pure, and act as a basic flux. To form sodium silicate requires $\frac{105.31}{59.96} = 1.756$ times as much sodium carbonate as silica.

9. Sodium Bicarbonate.—Sodium bicarbonate, $NaHCO_3$, is a basic flux that is reduced readily by heat to sodium carbonate:

$$\underset{\substack{\text{sodium}\\\text{bicarbonate}\\166.86}}{2NaHCO_3} + \text{heat} = \underset{\substack{\text{sodium}\\\text{carbonate}\\105.31}}{Na_2CO_3} + \underset{\substack{\text{carbon}\\\text{dioxide}\\43.67}}{CO_2} + \underset{\substack{\text{water}\\17.88}}{H_2O}$$

Sodium bicarbonate should be gently heated at first, otherwise the carbon dioxide and steam may blow out some of the fine ore. The bicarbonate is converted into carbonate, and according to the equation it requires $\frac{166.86}{105.31} = 1.584$ parts of bicarbonate to produce 1 part of carbonate. The carbonate fuses at about 800° C., and combines with alumina, lime, and silica to form slag. According to the equation given in Art. 8, it requires $\frac{166.86}{59.96} = 2.781$ times as much sodium bicarbonate as silica to form sodium silicate. The carbon dioxide evolved by the fusion of sodium carbonate or bicarbonate will oxidize sulphur, metallic iron, and zinc, which go into the slag. Acid sodium carbonate, $NaHCO_3$, does not absorb so much water as soda.

10. Potassium Carbonate.—Potassium carbonate, K_2CO_3, is a basic flux and unites with silica to form potassium silicate, K_2SiO_3. The chemical properties of potassium carbonate are similar to sodium carbonate, so that it may be substituted for the latter in assaying. As, however, 2.3 times more potassium carbonate than silica is required to form potassium silicate, and as the former is the more expensive, it is customary to use the sodium carbonate. Sodium and potassium carbonates when fused together form a double salt that is more fusible than either alone.

11. Potassium Bicarbonate.—Potassium bicarbonate, $KHCO_3$, like sodium bicarbonate, is also a basic flux, and produces practically the same reactions. Before use, it should be dried and freed from lumps. Heat converts the bicarbonate into carbonate, in which form it fuses with silica to form silicate of potassium, according to the equation:

$$\underset{\substack{\text{potassium}\\ \text{bicarbonate}\\ 198.80}}{2KHCO_3} + \text{heat} = \underset{\substack{\text{potassium}\\ \text{carbonate}\\ 137.25}}{K_2CO_3} + \underset{\substack{\text{carbon}\\ \text{dioxide}\\ 43.67}}{CO_2} + \underset{\substack{\text{water}\\ 17.88}}{H_2O}$$

The carbonate then unites with the silica to form potassium silicate, thus:

K_2CO_3 +	SiO_2 =	K_2OSiO_2 +	CO_2
potassium carbonate	silica	potassium silicate	carbon dioxide
137.25	59.96	153.54	43.67

To form potassium silicate requires 3.33 times as much potassium bicarbonate as silica. Potassium bicarbonate is a desulphurizer, oxidizing reagent, and basic flux.

12. Borax.—Borax, $Na_2B_4O_7$, is a very active, fusible, acid flux, and is a constituent of all stock fluxes. It forms fusible compounds with silica, and fluxes sulphides, arsenides, tellurides, lime, and metallic oxides; it is not, however, a desulphurizer or an oxidizing agent. Ordinary borax, $Na_2B_4O_7 \cdot 10H_2O$, contains more than 47.24 per cent. of water, which is given off very readily under the influence of heat, causing the borax to puff and swell, and rendering it liable, if used for assay purposes, to overflow the crucible. The water is, therefore, previously removed from the borax used for assaying, either by calcination (slow heating at a temperature slightly higher than the boiling point of water) or by fusion to a clear glass, in an iron or chalk-lined clay crucible, the melted borax being poured on a clean surface, and when cold pulverized, forming what is known as *borax glass*. Borax glass is largely used as a "cover" for both crucible and scorifier assays, a little being spread over the top of the charge before it is put into the furnace. This cover fuses before the main charge, and thus prevents some loss through the volatilization of litharge and certain volatile gold and silver ores, besides helping to start the fusion.

13. Sodium Chloride.—Sodium chloride, $NaCl$, or common salt, is used as a cover for crucible charges in gold and silver assays. To produce the best results, it should be pure and finely pulverized. It reacts with metal sulphides producing metal chlorides and sodium sulphides, thus:

FeS_2 +	$4NaCl$ =	$FeCl_2$ +	$2Na_2S$ +	Cl
ferric disulphide	salt	ferric chloride	sodium sulphide	chlorine
119.14	232.24	161.04	155.16	35.18

From this equation, it is seen that $\frac{232.24}{119.14} = 1.949$ times as much salt as ferric disulphide is required to desulphurize ferric sulphide or iron pyrite. Salt does not aid directly in the solution of silica; it does dissolve, however, the metal oxides, and prepares them to combine with silica. Salt will free lead from iron, copper, bismuth, arsenic, antimony, and metals that delay and sometimes spoil the cupellation process. Most of the metals when combined with chlorine are volatile, and in fire-assays, other than those of gold and silver, salt should not be used. Salt forms a liquid cover, prevents loss by ebullition, and washes the sides of the crucible.

14. Litharge.—Litharge, PbO, or yellow oxide of lead, acts as a basic flux and as an oxidizing and desulphurizing agent, and by reduction to a metal supplies the necessary lead for the collection of the gold and silver in crucible assays. Litharge is never entirely free from silver, and each new lot should be assayed, in order that the weight of the silver contained in the litharge of a crucible charge may be deducted from the weight of the resulting button, and the silver not credited to the ore. The crucible method of assay is used for determining the silver in litharge, and the charge taken is usually 1 or 2 A. T.; but the assayer will save time, trouble, and the possibility of arithmetical error if he uses for the assay charge the same amount of litharge as he uses in the flux for his gold-silver assay charges. A good charge for the litharge assay is: litharge, 2 A. T.; sodium bicarbonate, 1 A. T.; argol, 1 g. Cover with borax, and fuse as in the regular assay for gold and silver ores. Charcoal or flour may be used instead of argol. If the assayer mixes his litharge with his stock flux, he need only run duplicate assays of the flux alone, using the same amount for the charge as he mixes with his ore assays. A little silica (sand or powdered glass) added to the litharge-assay charge will save the crucible, which will otherwise be corroded to furnish the necessary silica for the slag.

It is claimed that red lead, Pb_3O_4, oxidizes silver, and thus causes loss; hence, litharge should be free from red lead. White lead, $PbCO_3$, or $2PbCO_3 \cdot PbH_2O_2$, is sometimes used in place of litharge. The first formula given for white lead is that of the native product, the second is that of the manufactured product known as *Dutch white*. Metallic oxides that are difficultly fusible alone are readily dissolved by lead oxide, with which they form a basic slag and attack the silica in the crucible. Lead and silica form a lead silicate, thus:

$$\underset{\substack{\text{lead}\\\text{oxide}\\221.23}}{PbO} + \underset{\substack{\text{silica}\\59.96}}{SiO_2} = \underset{\substack{\text{lead}\\\text{silicate}\\281.19}}{PbSiO_3}$$

From this equation it may be ascertained that it takes $\frac{221.23}{59.96} = 3.689$ parts of litharge to 1 part of silica by weight to form lead silicate, which is more fluid when fused than sodium silicate. Lead may form double fusible silicates —hence, silica bricks cannot be used for lead-furnace linings—but it has no action on lime or magnesia except in the presence of silicates and borates. As already stated, litharge is a desulphurizer, and in the proper proportions can be used for that purpose; it is, however, difficult to oxidize sulphides thoroughly in a crucible. The reaction said to occur is as follows:

$$\underset{\substack{\text{ferric}\\\text{sulphide}\\119.14}}{FeS_2} + \underset{\substack{\text{lead}\\\text{oxide}\\1{,}548.61}}{7PbO} = \underset{\substack{\text{ferrous}\\\text{oxide}\\71.38}}{FeO} + \underset{\substack{\text{sulphur}\\\text{trioxide}\\158.92}}{2SO_3} + \underset{\substack{\text{lead}\\1{,}437.45}}{7Pb}$$

This operation would require 12.997 g. of lead for every gram of ferric sulphide desulphurized, and is not so cheap an operation as desulphurizing with niter.

15. Silica.—Silica, SiO_2, is oxide of silicon, and is termed the acid constituent of rocks. When an ore to be assayed contains too much basic material it is necessary, in order to make a fusible slag, to add silica. Usually quartz sand or powdered glass—either ordinary window glass or

bottle glass (not plate glass, which contains lead)—is the material added as a flux in assaying.

Powdered quartz is a good flux, but lime glass, which is ordinary window glass, is the best unless the ore is very basic. The reaction that occurs may be expressed by the equation:

$$SiO_2 + CaCO_3 = CaSiO_3 + CO_2$$

silica	calcium carbonate	calcium silicate	carbon dioxide
59.96	99.25	115.54	43.67

It requires, therefore, $\frac{99.25}{59.96} = 1.655$ times as much limestone as silica to form calcium silicate, or if lime, CaO, is used, it will require $\frac{55.58}{59.96} = .927$ times as much lime as silica.

16. Potassium Nitrate.—Niter, or saltpeter, KNO_3, is a basic flux and a very powerful oxidizing and desulphurizing agent. Its use is, however, objectionable for various reasons. In the first place, its oxidizing power must be determined, as the amount used must be only just sufficient to accomplish the purpose for which it is added, any excess tending to prevent the reduction of the litharge. This determination involves two sets of assays. The reducing power of the stock flux must first be tested by running duplicate charges and weighing the resulting lead buttons. The amounts of flux and litharge in these charges should be the same as are used in a regular assay charge, the litharge being somewhat in excess of the amount the flux will reduce to metal. Two similar charges are run next, with the addition of 1 g. of niter to each. The buttons from this assay will be smaller than those from the previous one, and the difference between the average weight of the lead buttons from the two assays—without and with niter—represents the oxidizing power of niter per gram. If more than 1 g. of niter is used, the difference in weight of the buttons will have to be divided by the number of grams of niter used to obtain the oxidizing power per gram. After the oxidizing power of the niter has been determined, it is necessary, before the niter can be used in an assay charge, to determine the

reducing power of the ore with which it is to be used, in order to know just how much niter to add, and avoid excess. To do this, make up the following charge: ore, $\frac{1}{10}$ A. T.; litharge, 15 g.; sodium bicarbonate (or mixed soda and potassium bicarbonate), 10 g. Run this charge like the previous charges, and weigh the resulting button. The button reduced by $\frac{1}{2}$ A. T. of ore would be five times as heavy; and in an ordinary assay this weight would be added to that of the button reduced by the flux charge. The amount of niter added should be just sufficient to reduce the button to the desired weight. From the following equation the quantity of niter that will oxidize lead is found:

$$4Pb + 2KNO_3 = 4PbO + K_2O + NO + N$$

lead	niter	litharge	potassium oxide	nitrous oxide	nitrogen
821.40	200.84	884.92	93.58	29.81	13.93

This equation shows that 1 part of niter will oxidize 4 + parts of lead.

Besides necessitating all the extra work, niter in the flux is troublesome in itself. Its oxygen is given off so rapidly as to cause deflagration and spitting of the charges before they commence to melt, and, unless very large crucibles are used, charges containing niter are almost certain to boil over if left unwatched, as the niter causes violent boiling and effervescence. Taking all these things into consideration, most assayers prefer to use salt, litharge, iron wire, or nails rather than niter to prevent the sulphur and arsenic going into the buttons from sulphides and arsenides—the principal purpose for which niter is used.

17. Potassium Cyanide.—Potassium cyanide is a very powerful desulphurizing and reducing flux. It combines with oxygen, forming cyanates, thus:

$$PbO + KCN = Pb + KCNO$$

litharge	potassium cyanide	lead	potassium cyanate
221.23	64.69	205.35	80.57

According to this equation, 1 part by weight of potassium cyanide will reduce 3.4 parts by weight of litharge.

Potassium cyanide combines with sulphur to form thiocyanate, or potassium sulphocyanide, thus:

$$\underset{\substack{\text{lead}\\ \text{sulphide}\\ 237.17}}{PbS} + \underset{\substack{\text{potassium}\\ \text{cyanide}\\ 64.69}}{KCN} = \underset{\substack{\text{lead}\\ 205.35}}{Pb} + \underset{\substack{\text{thiocyanate}\\ 96.51}}{KSCN}$$

From this equation, it may be calculated that 1 part by weight of potassium cyanide will desulphurize 3.66 parts by weight of lead sulphide.

Potassium cyanide in lead flux is liable to reduce some of the more readily oxidizable metals—such as bismuth, copper, iron, tin, and antimony—along with the lead, causing a brittle, heavy button. Pure potassium cyanide is intensely poisonous, and should be handled with the greatest care. It should never be touched with cracked or sore hands, and should be ground in the open air, with a towel over the top of the mortar. For this reason, it is employed but little in assaying, the commercial cyanide being used instead. The ferrocyanide of potassium acts in a similar manner in the flux, though much less powerfully, and is much safer to handle, although care should be exercised in its case also. Like niter, both cyanide and ferrocyanide are liable to cause boiling over, and their place as desulphurizers is usually filled by iron wire or nails.

18. Iron, *Fe*.—Metallic iron is a powerful basic flux and desulphurizer. Its principal use in the crucible assay of sulphide and arsenide ores is to form a matte with the sulphur and arsenic, and thus keep them out of the lead button. From two to four wrought-iron cut nails, according to the amount of sulphur in the charge, are stuck point downwards into the crucible before putting it into the fire. As the charge melts, the sulphur rapidly eats away the iron, forming iron sulphide, which is dissolved by the slag if only a moderate quantity is present. If there is much of this iron sulphide, it forms a distinct layer of matte between the button and the slag, both before and after pouring. The matte can be readily distinguished from both slag and button by its crystalline structure and metallic luster. If any of the nails

remain undissolved in the crucible, they should be removed, before pouring, by means of the small crucible tongs, tapping them lightly against the edge of the crucible, as they are withdrawn, to shake off any adhering globules of lead.

19. Lead, *Pb*.—Metallic lead acts as a basic flux and also as a collector of the precious metals in gold and silver assays. Test lead (pure granulated lead) is the principal flux used in the scorification assay. Sheet lead, or lead foil, is used in assaying bullion. Test lead, like litharge, almost invariably contains more or less silver, and should be assayed for that metal, and the proper deduction made from the results of all silver assays in which it is used. The assay is run exactly like an ordinary scorification assay, adding silica to the charge.

20. Other Fluxes.—Powdered lime, CaO, powdered magnesia, MgO, fluor spar, CaF_2, and cryolite, $3NaF \cdot AlF_3$, are basic fluxes that are found useful in special cases.

REDUCERS

21. Reducing Agents.—It has been stated that potassium cyanide is a powerful oxidizer and reducer, and is also a flux. The function of a flux is to slag the impurities in an ore so that the metal oxide may be collected by the litharge, and both be reduced by some carbonaceous matter in the mixture. There is carbon in potassium cyanide, which accounts for its reducing properties; but, on account of its poisonous properties, it is very little used as a reducer. A reducing agent is a substance that will reduce an oxide from a higher to a lower state of oxidization, or remove the oxygen entirely and reduce the oxide to a metal.

22. Charcoal.—Charcoal absorbs moisture and gases from the atmosphere, for which reason its reducing power will vary and should be determined. For this purpose 1 g. of charcoal, 2 A. T. of litharge, and ½ A. T. of soda bicarbonate are thoroughly mixed, placed in a crucible, and fused in a hot fire. After cooling, the button resulting from

the fusion is weighed, and this weight represents what 1 g. of the charcoal will reduce. Usually, good charcoal will reduce from 20 to 30 parts of lead.

The charcoal for assaying must be finely pulverized. In burning, it is converted into carbon dioxide, CO_2, and this coming in contact with incandescent metallic oxides is broken up and converted into carbon monoxide, CO, and oxygen. Carbon monoxide is a reducing gas; that is, it absorbs the oxygen from metallic oxides and converts the latter into metals.

23. Flour.—Flour is a substance that contains carbon. In the heat of the assay furnace it is converted into charcoal, then into carbon dioxide, and finally into carbon monoxide, in which form it reduces the oxides of metals into metals. One part of flour will reduce about 15 parts of lead from lead oxide. Flour is not so good a reducer as charcoal, because it must be converted into charcoal before the reducing action takes place, and moreover contains substances that reduce its carbonizing effect, and make it impure carbon.

24. Starch, Sugar, and Gum.—Starch, sugar, and gum are carbonaceous materials, and can be employed as reducing agents. One part of undried starch will reduce 11 parts of lead. One part of dried starch will reduce 13 parts of lead. One part of sugar will reduce 14½ parts of lead. One part of gum arabic will reduce 11 parts of lead.

25. Argol.—Argol is impure cream of tartar or potassium bitartrate, $KHC_4H_4O_6$, and has more carbon, and therefore greater reducing power, than the pure compound. Heat changes argol to potassium carbonate, K_2CO_3, and carbon, the water being driven off. It acts as a basic flux as well as a reducer, owing to the potassium it contains. To determine the reducing power of argol, it should be pulverized dry and mixed in the following proportions:

Argol	2 g.
Litharge	2 A. T.
Soda bicarbonate	½ A. T.

This mixture is fused in a crucible in a hot fire, and when cool the button is extracted and its weight in grams ascertained. This weight divided by 2, the number of grams of argol taken, will give the weight of the lead 1 g. of argol will reduce from litharge. Approximately, 1 part of argol by weight should reduce about 6 parts of lead by weight.

26. Reducers in General.—Carbonaceous materials may be used as reducers. Sulphur, arsenic, and antimony in ores have a reducing effect and assist the reducer in the flux. This fact should be kept in mind in making up the flux for any particular ore. Powdered sulphur is sometimes, though seldom, used as a reducing reagent in flux.

TABLE III

APPROXIMATE REDUCING POWER OF REDUCING AGENTS

(Expressed in terms of parts of metallic lead reduced from litharge by 1 part of the reducer)

Reducing Agent 1 Part	Amount of Lead Reduced Parts
Charcoal	20 to 30
Hard coal	25
Coke	24
Soft coal	22
Wheat flour	15
White sugar	14½
Starch	11½ to 13
Gum arabic	11
Crude argol	5½ to 8½
Cream of tartar	4½ to 6½

27. Table III gives the approximate reducing power of such reducing reagents as are commonly used in assaying, in terms of the number of parts of lead reduced from litharge by 1 part of the reducer. These figures are, however, only *approximate*, and should not be used in the determination of the oxidizing or reducing power of an ore

or a reagent. For this purpose, a test assay of the reducer should always be run, following the method given for determining the reducing power of charcoal and argol and using from $\frac{1}{2}$ g. to 2 g. of the reducer in place of the ore charge. For the calculation of general charges, however, they are sufficiently close.

28. Sulphide Ores.—A large number of ores contain impurities such as antimony, arsenic, selenium, sulphur, and tellurium, or a mixture of one or more of these non-metallic elements, usually with sulphur predominating. Ores of this description are not so readily assayed as silicious ores; in fact, they act as reducers, thus increasing the size of the button beyond 20 g., or else produce a hard or brittle button that must be scorified. In some cases, a matte or speiss will be formed, owing to the presence of sulphides, etc., and this will hold some of the gold and silver, particularly if any copper is present in the ore. Hard buttons are caused by such elements as iron, copper, nickel, platinum, or a large quantity of silver.

Brittle buttons are caused by the non-metallic elements, or by zinc, gold, and platinum in large amounts. Owing to the difficulties mentioned, sulphide ores, previous to assaying, are given special treatment or else are roasted. Sulphide ores are given a preliminary assay, or the fluxes used are gauged from experience, and added to the ore. The assay of sulphide ores will be found under the heading Refractory Ores.

29. Preliminary Assaying.—To find the reducing power of an arsenical or sulphide ore, and to regulate this power, a preliminary assay is advisable. For this purpose take

5 g. of the ore pulp
80 g. of litharge
20 g. of soda bicarbonate
5 g. of borax
5 g. of silica
Borax cover.

When the fusion is complete, treat the lead button in the ordinary way.

If no lead has been reduced, the ore has no reducing power. If less than 3 g. of lead are reduced, a button from an assay ton would be less than $3 \times 6 = 18$ g., the quantity of pulp in this case being practically $\frac{5}{30}$ or $\frac{1}{6}$ of an assay ton, or $\frac{29.166}{6}$ g. In order to bring up the button to the 20-g. size, argol or charcoal must be added as a reducer.

If $3\frac{1}{4}$ g. of lead is reduced, the button will be about right, as it will weigh $3\frac{1}{4} \times 6 = 19.5$ g. for an assay ton of ore. If the button weighed 5 g., then the button for an assay ton would weigh 30 g., or 10 g. more than is advisable, and niter must be added to oxidize this excess. It has been demonstrated that 1 part of niter will oxidize about 4 parts of lead; hence, $\frac{10}{4} = 2.5$ g. of niter would need to be added to the regular charge. Sometimes, the niter calculated for a preliminary assay does not produce the desired results and very little lead is reduced, owing to iron oxide being in the pulp. This may be corrected by the addition of more niter or a large quantity of litharge.

30. Oxidized Ores.—Ores that contain oxides of iron, copper, and manganese are, like sulphide ores, more difficult to assay than silicious ores. The oxides of the base metals make the lead button too small or else prevent it from forming when the flux is for a silicious ore. This is due to their oxidizing power and to their uniting with the litharge. Usually, an oxidized ore can be recognized by its reddish or brown color—but not always, since there are the black copper oxide melaconite; the black manganese oxides, pislomelane and pyrolusite; the black iron oxide, magnetite, and sometimes hematite; and the titanium mineral ilmenite. To correct the oxidizing power of such ores a reducer is required, and to ascertain the quantity of reducer for the flux, a preliminary assay is made.

31. Preliminary Assay for Oxide Ores.—To determine the oxidizing power of an ore, take the following

charge and fuse it in a crucible; then, treat the button as directed:

5 g. of ore
40 g. of litharge
10 g. of sodium bicarbonate
3 g. of argol or $1\frac{1}{2}$ g. of flour
5 g. of borax glass
Salt cover.

Assume that the argol used is capable of reducing 6 parts by weight of litharge, but that 5 g. of ore with 3 g. of argol gave a button weighing 15 g. It is evident, then, that 5 g. of ore oxidized $18 - 15 = 3$ g. of lead, and that an assay ton, which is practically 30 g., would oxidize 18 g. of lead. To neutralize this oxidizing effect, it will require $\frac{18}{15} = 1.2$ g. of argol, and to furnish a button of 20 g. from an assay ton will require 1.33 g. of argol additional.

SLAGS

32. Mixed Fluxes.—Assayers whose work is confined to ores in one district, soon learn to mix a flux suitable to those ores. Public assayers have to deal with a variety of ores from many districts, and should roughly calculate the fluxes for each ore. Experienced assayers with the aid of a microscope can approximate the necessary fluxes that, when mixed with an ore, will make a fusible slag.

In case of doubt, a qualitative test of the ore will furnish data sufficient for the calculation of a suitable slag, or will indicate a stock flux suitable to the ore. Certain fluxes have been handed down from generation to generation, and while they will flux the greater number of silver and gold ores they are not entirely universal. These are termed **stock fluxes.**

33. Fluidity of Slags.—Assayers that fail to formulate slags with a fluidity sufficient to permit the collecting metal to circulate freely cannot obtain correct results. When slags are pasty, the precious metals are unable to alloy with the lead and are held in suspension. Fusions of this description are discarded. While the object of the assayer is to

produce a fluid slag, nevertheless he must not have so fluid a mixture that the collecting metal will sink before alloying with the gold and silver, as in this case the precious metals may remain incorporated in the slag. In order to form suitable slags, the silica in the gangue must be balanced with the bases of the gangue, and the excess of silica in the ore fluxed with litharge, soda, borax, etc.

34. Silicates.—Silica combines with bases in widely varying proportions to form silicates. The fusibility of silicates depends on the bases and the percentage of silica they contain. Silicates are classified according to the ratio of the oxygen in the base to the oxygen in the silica. In the general formulas given, $\ddot{R}$ represents the base, and may be lime, sodium, or other metallic oxide:

Name	Formulas	Oxygen Ratio
Subsilicate	$4\ddot{R}O \cdot SiO_2$	2 : 1
Monosilicate	$2\ddot{R}O \cdot SiO_2$	1 : 1
Bisilicate	$\ddot{R}O \cdot SiO_2$	1 : 2
Trisilicate	$2\ddot{R}O \cdot 3SiO_2$	1 : 3
Sesquisilicate	$4\ddot{R}O \cdot 3SiO_2$	2 : 3

Bases that had the form of alumina, Al_2O_3, would have the formula $\dddot{R}_2$ and be written as follows:

Name	Formulas	Oxygen Ratio
Subsilicate	$4Al_2O_3 \cdot 3SiO_2$	12 : 6 or 2 : 1
Monosilicate	$2Al_2O_3 \cdot 3SiO_2$	6 : 6 or 1 : 1
Bisilicate	$Al_2O_3 \cdot 3SiO_2$	3 : 6 or 1 : 2
Trisilicate	$2Al_2O_3 \cdot 9SiO_2$	6 : 18 or 1 : 3
Sesquisilicate	$4Al_2O_3 \cdot 9SiO_2$	12 : 18 or 2 : 3

Some silicates are more fusible than others, but, for assays in general, monosilicate or sesquisilicate slags—and at times a bisilicate—are preferable. When a bisilicate slag is to be calculated, the silica required for a monosilicate is calculated and multiplied by 2; for a trisilicate slag, it would be multiplied by 3; but for a subsilicate it would be divided by 2. When the base of a monosilicate slag has been calculated and another silicate is to be formed, the bases must be

divided by 2 for a bisilicate; by 3 for a trisilicate; by $\frac{3}{2}$ for a sesquisilicate; and be multiplied by 2 for a subsilicate.

35. Calcium Silicate.—The two most fusible silicates of calcium are the bisilicate, $CaO{\cdot}SiO_2$, and the sesquisilicate, $4CaO{\cdot}3SiO_2$. The assayer rarely adds CaO to an assay charge, but in the form of $CaCO_3$ it is frequently present in ores as a common gangue mineral. Before a silicate of calcium can be formed, the carbon dioxide of the limestone $CaCO_3$ must be expelled by heat, according to the reaction:

$$\underset{\substack{\text{limestone}\\198.50}}{2CaCO_3} + \text{heat} = \underset{\substack{\text{lime}\\111.16}}{2CaO} + \underset{\substack{\text{carbon}\\\text{dioxide}\\87.34}}{2CO_2}$$

From this equation, it requires $\frac{198.50}{111.16} = 1.785$ times as much limestone as lime to form lime.

The formula for calcium monosilicate is $2CaO{\cdot}SiO_2$, and is obtained as follows:

$$\underset{\substack{\text{lime}\\111.16}}{2CaO} + \underset{\substack{\text{silica}\\59.96}}{SiO_2} = \underset{\substack{\text{calcium}\\\text{monosilicate}\\171.12}}{2CaO{\cdot}SiO_2}$$

From this equation, it requires $\frac{111.16}{59.96} = 1.853$ times as much lime as silica to form calcium silicate, or $\frac{198.50}{59.96}$ $= 3.3$ times as much limestone as silica. The quantity of silica necessary to flux limestone is $\frac{59.96}{198.50} = .302$, and the quantity of silica for fluxing lime is $\frac{59.96}{111.16} = .539$. This latter factor will be found useful in slag calculations, as, when the percentage of limestone is known, the weight of silica to form a monosilicate can be found by multiplying by the factor .302; or, if the percentage of lime is known in an ore, the factor .539 is used for a monosilicate.

Silicates of lime are not so fusible as silicates of lead, potassium, or sodium, so that lime is not used as a flux in assaying.

36. Magnesium Silicate.—Magnesia carbonate, $MgCO_3$, and magnesium oxide, MgO, are infusible alone, but when combined with silica form a series of silicates. The most fusible of these silicates are the trisilicate, $2MgO \cdot 3SiO_2$, and the sesquisilicate, $4MgO \cdot 3SiO_2$. As in the case of calcium carbonate, magnesium limestone is found as a gangue rock, but before the magnesium carbonate can form a silicate it must be converted into magnesium oxide by heat. The reactions that occur in the formation of a monosilicate slag are:

$$\underset{\substack{\text{magnesium}\\\text{carbonate}\\167.46}}{2MgCO_3} + \text{heat} = \underset{\substack{\text{magnesium}\\\text{oxide}\\80.12}}{2MgO} + \underset{\substack{\text{carbon}\\\text{dioxide}\\87.34}}{2CO_2}$$

$$\underset{\substack{\text{magnesium}\\\text{oxide}\\80.12}}{2MgO} + \underset{\substack{\text{silica}\\59.96}}{SiO_2} = \underset{\substack{\text{magnesium}\\\text{silicate}\\140.08}}{2MgO \cdot SiO_2}$$

The quantity of silica necessary to form a magnesium monosilicate is, when factored for magnesium carbonate, $\frac{59.96}{167.46} = .358$, and when factored for magnesia, .758. From the equations, 1 part by weight of silica requires $\frac{167.46}{59.96} = 2.792$ parts of magnesium carbonate, and 1 part by weight of silica requires $\frac{80.12}{59.96} = 1.336$ parts of magnesia.

37. Aluminum Silicate.—The temperature of the assay furnace is not sufficient to form an aluminum silicate. From the fact that double silicates are more fusible than single silicates, and that double silicates are the ones usually formed, the calculations are made for alumina, Al_2O_3, as if it was fusible as a monosilicate. The formula for a monosilicate is $2Al_2O_3 \cdot 3SiO_2$, from which it is found that 1 part of silica requires $\frac{202.88}{179.88} = 1.122$ parts of alumina for the slag, and that the factor for alumina, or the necessary amount of silica, is $\frac{179.88}{202.88} = .886$.

38. Barium Silicate.—Barium is found in gangue mineral in the carbonate and sulphate form. It is usually associated with lead ores. The two most fusible silicates are said to have the formulas $BaO \cdot 3SiO_2$ and $BaO \cdot 4SiO_2$; however, since barium is not alone, but is associated with other silicates, it may be calculated as a monosilicate having the formula $2BaO \cdot SiO_2$. This formula requires that for 1 part of silica 5.08 parts of barium oxide be taken. The silica factor for barium oxide in a monosilicate slag is $\frac{59.96}{304.56} = .196$.

39. Potassium Silicate.—The basic potassium silicates are the most fusible, but, owing to their fusibility and to the fact that they attack the crucible, the monosilicate is taken for assay charges. Usually, potassium bicarbonate or potassium carbonate are the fluxes used to obtain the silicate desired, and as they contain carbon dioxide, which heat expels, the reactions are similar to those calculated for calcium carbonate:

$$\underset{\substack{\text{bicarbonate}\\\text{soda}\\397.60}}{4KHCO_3} + \text{heat} = \underset{\substack{\text{potassium}\\\text{oxide}\\187.16}}{2K_2O} + \underset{\substack{\text{carbon}\\\text{dioxide}\\174.68}}{4CO_2} + \underset{\substack{\text{water}\\35.76}}{2H_2O}$$

$$\underset{\substack{\text{potassium}\\\text{oxide}\\187.16}}{2K_2O} + \underset{\substack{\text{silica}\\59.96}}{SiO_2} = \underset{\substack{\text{potassium}\\\text{monosilicate}\\247.12}}{2K_2O \cdot SiO_2}$$

For each part of silica it requires $\frac{187.16}{59.96} = 3.12$ parts of K_2O, or $\frac{397.60}{59.96} = 6.63$ parts of $KHCO_3$.

When the parts of potassium oxide in an ore are found they are to be multiplied by the silica factor $\frac{59.96}{187.16} = .320$.

40. Sodium Silicate.—Sodium subsilicates are the most fusible, but a monosilicate of soda is used in assaying. A moderately high formation temperature does not prevent the reduction of the metals; in fact, in some smelting

operations, it is necessary. In assaying, therefore, the monosilicate slag is calculated. The Na_2O for the slag is furnished as sodium bicarbonate, according to the equation:

$$\underset{\substack{\text{sodium}\\\text{bicarbonate}\\333.72}}{4NaHCO_3} + \text{Heat} = \underset{\substack{\text{sodium}\\\text{oxide}\\123.28}}{2Na_2O} + \underset{\substack{\text{carbon}\\\text{dioxide}\\174.68}}{4CO_2} + \underset{\substack{\text{water}\\35.76}}{2H_2O}$$

The monosilicate has the formula $2Na_2O \cdot SiO_2$, hence, it requires $\frac{123.28}{59.96} = 2.05$ times as much sodium oxide as silica, and $\frac{333.72}{59.96} = 5.56$ times as much as sodium bicarbonate as silica. The factor for silica for a monosilicate will be $\frac{59.96}{123.28} = .486$. The percentage of sodium oxide in sodium bicarbonate is $\frac{123.28 \times 100}{333.72} = 36.94$.

41. Lead Silicate.—As lead bisilicate has the formula $PbO \cdot SiO_2$, and is an easily fusible silicate; the monosilicate, however, is the one adopted for calculating assay slags. The monosilicate, $2PbO \cdot SiO_2$, is derived as follows:

$$\underset{\substack{\text{litharge}\\442.46}}{2PbO} + \underset{\substack{\text{silica}\\59.96}}{SiO_2} = \underset{\substack{\text{lead monosilicate}\\502.42}}{2PbO \cdot SiO_2}$$

It requires $\frac{442.46}{59.96} = 7.37$ times as much litharge as silica to form a monosilicate, and the factor for silica is $\frac{59.96}{442.46} = .135$.

42. Ferrous Silicate.—The subsilicate of iron is said to make the most fusible silicate, but in the following calculation the monosilicate is adopted. The formula for the monosilicate is $2FeO \cdot SiO_2$, from which it will be found that 1 part of silica requires 2.38 parts of ferrous oxide, and that the silica factor is .419. The ferrous oxide, FeO, is furnished by ferric oxide, Fe_2O_3, found in the ore, and is not added to the assay charge as a flux. Ferric oxide consists of 90 per cent. FeO and 10 per cent. oxygen.

Before the ferric oxide is reduced to ferrous oxide, a reaction probably like that given below takes place:

$$\underset{\substack{\text{ferric oxide}\\317.28}}{2Fe_2O_3} + \underset{\substack{\text{carbon}\\11.91}}{C} = \underset{\substack{\text{ferrous oxide}\\285.52}}{4FeO} + \underset{\substack{\text{carbon dioxide}\\43.67}}{CO_2}$$

One gram of ferric oxide will require $\frac{11.91}{317.28} = .037$ g. of carbon to reduce it to FeO. Charcoal is the nearest approach to pure carbon that the assayer has at command.

43. Copper Silicate.—The formula $CuO{\cdot}SiO_2$ corresponds to the mineral chrysocolla; it would, however, be proper in the case of copper to use a monosilicate having the formula $2CuO{\cdot}SiO_2$. Since the molecular weight of $2CuO$ is 157.96, it will require $\frac{157.96}{59.96} = 2.63$ parts of copper oxide for each part of silica. The factor for ascertaining the percentage of silica required to flux any given quantity of copper oxide is .379.

44. Typical Assay Slags.—A slag of low formation point (590° C.) and considerable viscosity may be had from the mixture $Na_2O{\cdot}PbO{\cdot}4SiO_2{\cdot}2B_2O_3$, which may be written $PbO + 4SiO_2 + Na_2B_4O_7$, the latter being sodium borate, or borax glass. By calculating from the atomic weights, the following charge will yield this slag:

$$\text{Litharge, } PbO,\ 221.23 \text{ and } \frac{221.23 \times 100}{661.59} = 33.4 \text{ g.}$$

$$\text{Silica, } 4SiO_2,\ 239.84 \text{ and } \frac{239.84 \times 100}{661.59} = 36.2 \text{ g.}$$

$$\text{Borax, } Na_2B_4O_7,\ \frac{200.52}{661.59} \text{ and } \frac{200.52 \times 100}{661.59} = 30.3 \text{ g.}$$

Another slag with a melting point of 740° C., and a desirable one for aluminous ores is

$$Na_2O{\cdot}PbO{\cdot}Al_2O_3{\cdot}6SiO_2{\cdot}2B_2O_3$$

This may be written so as to combine the sodium and borax into sodium borate, $Na_2B_4O_7$. Then, by calculation

from the atomic weights, the percentage weights to be added to an assay charge are found to be as follows:

$$\text{Borax glass, } Na_2B_4O_7,\ \frac{200.52 \times 100}{882.95} = 22.7 \text{ per cent.}$$

$$\text{Litharge, } PbO,\ \frac{221.23 \times 100}{882.95} = 25.05 \text{ per cent.}$$

$$\text{Alumina, } Al_2O_3,\ \frac{101.44 \times 100}{882.95} = 11.4 \text{ per cent.}$$

$$\text{Silica, } SiO_2,\ \frac{359.76 \times 100}{882.95} = 40.7 \text{ per cent.}$$

45. Balling's Table.—One or more of the bases FeO, CaO, MgO, MnO, BaO, and Al_2O_3 are present in nearly all ores, and in all probability SiO_2 is present in every ore. In assay work, it is customary to add PbO and Na_2O to lower the formation point of the slag. Table IV was, with a few exceptions, arranged by Balling to simplify slag calculations. The table is calculated on unit molecular base ratios; for instance, the first horizontal top line is calculated with sodium oxide as the unit, and the litharge factor is found by dividing the molecular weight of litharge by the molecular weight of sodium oxide. In the second horizontal line litharge is taken as the unit, and the other oxides are calculated by dividing their molecular weights by the molecular weight of litharge.

46. Calculation of an Assay Slag.—It is desired to calculate the charge that will produce a monosilicate slag having the composition $Na_2O{\cdot}PbO{\cdot}FeO{\cdot}CaO{\cdot}2SiO_2$, using as the unit Na_2O and the parts of the other bases given in Table IV, and corresponding to sodium oxide as unity.

The silica required will be found as follows:

$$\begin{array}{lrcl}
Na_2O, & 1 & \times .486 = & .486 \\
PbO, & 3.59 & \times .136 = & .486 \\
FeO, & 1.16 & \times .419 = & .486 \\
CaO, & .903 & \times .539 = & \underline{.486} \\
 & & & 1.944
\end{array}$$

The silica can be determined by figuring it for one base and multiplying that figure by the number of oxygen molecules present in the bases. As stated in Art. 40, the

TABLE IV

THE CALCULATION OF SLAGS

One Part by Weight of the Base	Parts of Other Bases Necessary										One Part SiO_2 Requires to Form a Monosilicate	Parts of Silica Necessary for Monosilicate
	Na_2O	PbO	K_2O	CaO	MgO	Al_2O_3	FeO	CuO	ZnO	MnO		
Sodium oxide . . .	1.000	3.591	1.518	.903	.645	1.645	1.158	1.281	1.311	1.136	2.06 parts Na_2O	.486
Litharge	.279	1.000	.423	.252	.181	.458	.322	.356	.365	.318	7.38 parts PbO	.136
Ferrous oxide . . .	.863	3.105	1.311	.779	.568	1.420	1.000	1.106	1.132	.987	2.38 parts FeO	.419
Potassium oxide .	.659	2.365	1.000	.594	.428	1.083	.763	.844	.863	.753	3.12 parts K_2O	.319
Calcium oxide . .	1.109	3.981	1.684	1.000	.720	1.829	1.284	1.420	1.453	1.268	1.85 parts CaO	.539
Magnesium oxide .	1.539	5.524	2.336	1.388	1.000	2.531	1.782	1.972	2.016	1.756	1.33 parts MgO	.748
Alumina	.608	2.182	.923	.548	.395	1.000	.704	.778	.797	.695	1.16 parts Al_2O_3	.295
Cupric oxide, CuO	.780	2.802	1.185	.704	.507	1.284	.903	1.000	1.023	.892	2.63 parts CuO	.378
Zinc oxide	.763	2.738	1.158	.688	.496	1.255	.883	.977	1.000	.872	2.69 parts ZnO	.371
Manganous oxide .	.873	3.131	1.324	.788	.563	1.436	1.014	1.101	1.140	1.000	2.36 parts MnO	.422

sodium oxide for assay charges is obtained from bicarbonate of soda, which contains about 37 per cent. of Na_2O; hence, $\frac{1 \times 100}{37} = 2.7$ parts of $NaHCO_3$ will be needed. As a usual thing, ferrous oxide is derived from ferric oxide, which it is assumed contains 80 per cent. of Fe_2O_3, and 17 per cent. of SiO_2. From Table IV, 1.16 parts FeO are required, and since Fe_2O_3 consists of 90 per cent. of FeO and 10 per cent. of O, the necessary ore to furnish the ferrous oxide is $\frac{1.16}{.90 \times .80} = 1.61$ parts.

The coal used has a reducing power of 20 parts of lead per part of coal; and in Art. 42 it is shown that it requires .037 part of coal to reduce 1 part of Fe_2O_3. The coal required for the iron is $1.61 \times .037 = .06$ part. Lead oxide contains 92 per cent. of lead, and it is desired to obtain a 20-g. lead button; hence, in addition to the 3.59 parts for the silicate, there must be added to the charge $\frac{20 \times 100}{90} = 22$ parts of litharge, or a total of 25.59 parts of litharge. The coal used will reduce 20 g. of lead for every gram of coal.

The lime is derived from calcite 98 per cent. pure, and this contains approximately 54 per cent. of CaO; therefore, $\frac{.903 \times 100}{54} = 1.67$ parts of calcite will be required. The iron ore contains 17 per cent. of silica, and this must be deducted from silica; thus, $1.61 \times .17 = .274$, and $1.944 - .274 = 1.67$ parts of silica are to be added to the charge. Assuming that each part represents a gram, the correct charge will be: $NaHCO_3$, 2.70 g.; PbO, 25.59 g.; Fe_2O_3, 1.61 g.; $CaCO_3$, 1.67 g.; SiO_2, 1.67 g.; charcoal, 1.06 g.

Let it be assumed that an assay slag of the composition $Na_2O{\cdot}PbO{\cdot}FeO{\cdot}CaO{\cdot}2SiO_2$ is to be calculated for an ore containing 95 per cent. of SiO_2, taking as a basis 1 A. T. ore 29.166 g., or, say, 30 g. An assay ton of the above ore will contain 28.5 g. of silica, which is divided into 4 parts to satisfy the bases present, thus: $\frac{28.5}{4} = 7.1$ g., and this will

go to such an amount of each base as will form a monosilicate. Thus:

7.1 g. of SiO_2 requires $7.1 \times 2.07 = 14.7$ g. of Na_2O
7.1 g. of SiO_2 requires $7.1 \times 7.36 = 52.25$ g. of PbO
7.1 g. of SiO_2 requires $7.1 \times 2.40 = 17.04$ g. of FeO
7.1 g. of SiO_2 requires $7.1 \times 1.86 = 13.20$ g. of CaO

The bicarbonate of soda required is $\frac{14.7 \times 100}{37} = 39.6$ g. The litharge required is $52.25 + 22 = 74.25$ g. for a 20-g. button. Assuming in this case that the iron is derived from siderite, $FeCO_3$, then the percentage of FeO in a pure iron carbonate is 62; hence, $\frac{17.04 \times 100}{62} = 27$ g. of $FeCO_3$ is required. Assuming that the limestone is 98 per cent. pure, then $\frac{13.2 \times 100}{54} = 24.4$ g. of limestone.

The complete charge for 1 A. T. of ore will be as follows: $CaCO_3$, 24.4 g.; $FeCO_3$, 27.0 g.; PbO, 74.25 g.; $NaHCO_3$, 39.60 g.; coal, 1.00 g.

47. In the two cases just given, the same slag was produced for a basic and a silicious ore, which brings out the fact that fluxes are always added of such nature and in such quantity as the ore demands to produce a slag of fairly constant composition. In the examples, the slag was limited to four bases, and the bases were present in unit molecular ratio; however, where an ore contains numerous bases, it is evident that they are not present in the unit molecular ratio, so that the formula of the slag would have the general form

$$(aPbO, bNa_2O, cFeO, dMgO, eAl_2O_3)\,x(SiO_2)$$

in which for a monosilicate

$$a + b + c + d + 3e = 2x$$

In order to obtain a slag of comparatively low formation temperature, the less fusible bases CaO, MgO, and Al_2O_3 must be in smaller proportions than those of the more fusible bases PbO, Na_2O, and FeO.

In assay practice, the fluxes added to an ore charge are limited to litharge, sodium oxide, and potassium oxide, so that when a silicious ore is assayed the slag approximates a monosilicate and borate of litharge and soda.

48. Fluxes for Quartzite.—A quartzite gangue rock gave the following analysis. If the per cent. of each base is multiplied by the silica factor in Table IV, the quantity of silica it will flux is found.

	Per Cent.
Silica, SiO_2	77
Lime, CaO	.7
Potassium oxide, K_2O	4.3
Magnesia, MgO	1.0
Alumina, Al_2O_3	13
Ferric oxide, Fe_2O_3	1.1
Sodium oxide, Na_2O	.7

Base	Per Cent.
Al_2O_3, 13 × .881	11.453
CaO, .7 × .539	.377
Fe_2O_3, 1.1 × .90 × .419	.415
K_2O, 4.3 × .319	1.372
Na_2O, .7 × .486	.340
MgO, 1.0 × .758	.758
Total,	14.715

The bases will slag 14.715 per cent. of the silica in the rock, leaving an excess of 77 − 14.715 = 62.285 per cent. to be provided for by the fluxes, soda, and litharge.

If an assay ton, or 30 g., of ore is taken, then 30 × .77 = 23.1 g. of silica is in the ore, from which 23.1 × 14.7 = 3.395 g. of silica is to be subtracted, leaving 19.705 g. to be fluxed by sodium oxide and litharge, or $\frac{19.70}{2} = 9.85$ g. for each.

$\frac{9.85 \times 2.07 \times 100}{37} = 55.1$ g. of $NaHCO_3$. 9.85 × 7.36 = 72.5 g. of PbO, which added to the litharge for a 20-g. button gives a total of 94.5 g. of PbO. The charge for a monosilicate slag would be:

ORE	1 A.T.
$NaHCO_3$	55.1 g.
PbO	94.5 g.
Coal	.1 g.
Borax glass cover.	

49. Color of Slags.—Slags will vary from a light green to black, according to the proportion of ferrous silicate in them. When copper predominates, a red slag of cuprous silicate is obtained. Manganese in excess will give a dark slag showing a violet color on thin edges by transmitted light. Lime gives a gray slag, as does also magnesia and zinc when iron is absent. When lead silicate predominates, it gives the slag a yellow color.

50. Use of Borax.—Borax has a solvent power on silicates and washes the sides of the crucible; it also acts as an acid flux. When 1 g. of borax is used, it should be fluxed with 1 g. of litharge or .5 g. of sodium carbonate. The borax glass cover helps keep the slag fluid and prevents the boiling mass from sticking to the sides of the crucible.

51. Fulton's Slags.—The following slags are readily calculated and made up:

1. $PbO{\cdot}Na_2O{\cdot}SiO_2$, monosilicate.
2. $PbO{\cdot}Na_2O{\cdot}2SiO_2$, bisilicate.
3. $2(PbO{\cdot}Na_2O{\cdot}CaO)3SiO_2$, monosilicate.
4. $2(2PbO{\cdot}2Na_2O{\cdot}CaO)10SiO_2$, bisilicate.
5. $2(FeO{\cdot}PbO{\cdot}Na_2O)2SiO_2{\cdot}4B_2O_3$, subsilicate.
6. $(2FeO{\cdot}2PbO{\cdot}CaO)3SiO_2$, sesquisilicate.

The charge for a monosilicate of lead would be:

Quartz ore	.5 A. T.
$NaHCO_3$	39 g.
PbO	55 g.
Borax glass cover.	

The charge for a sesquisilicate would be:

Silica ore	.5 A. T.
$NaHCO_3$	30 g.
PbO	38 g.
Coal	1 g.
Borax glass cover.	

The charge for a bisilicate would be:

Silica ore	.5 A. T.
$NaHCO_3$	20 g.
PbO	28 g.
Coal	1 g.
Borax glass cover.	

For a 20-g. lead button, 22 to 25 g. of litharge is to be added to the charges given.

MIXED FLUXES

52. The following are some of the formulas for stock crucible fluxes recommended by different authorities. These fluxes have all been thoroughly tested, and while no one of them is suited to *all* ores, any of them will flux the majority of ores met with in custom assay practice. Special fluxes, adapted to particular types of ores, are given in Art. **56.** An assayer whose work is mainly confined to ores of any particular type or district should calculate his flux or experiment with various fluxes until he finds the one best suited to those ores and should then stick to that as his stock flux.

53. Lead Fluxes.—The following fluxes are primarily calculated for the fire-assay of lead ores. They are all good general fluxes, however, and any one may be used as the basis of a gold-silver crucible flux, merely adding litharge.

No. 1.	Sodium bicarbonate	4 parts*
	Potassium carbonate	4 parts
	Borax glass	2 parts
	Flour	1 part
	Salt cover.	
No. 2.	Sodium bicarbonate	13 parts
	Potassium carbonate	10 parts
	Borax	5 parts
	Flour	2½ to 4 parts
	Salt cover.	

*In all the fluxes given in this treatise, the proportions of the constituents are given in parts by weight.

If the ore contains sulphur, the proportion of flour may be reduced, or for heavy sulphides the flour may be omitted entirely. From 1 to 4 tenpenny wrought-iron cut nails should be added to the charge for a sulphide before the salt or borax cover, or the proper amount of desulphurizer added.

54. Gold and Silver Crucible Fluxes.—Most of the gold-silver crucible fluxes are merely lead fluxes to which litharge has been added. The amount of litharge added should be 15 g. to give a lead button weighing about 10 g. from a ½-A.-T. assay. Any unreduced litharge acts as a flux and goes into slag. If a large amount of litharge is used, the reducing power of the charge must be kept down, so that too large a lead button will not be obtained. A good charge to use is 25 g. of litharge, as there is only a slight excess of litharge over the amount necessary to produce a 20-g. button, and unless the ore contains lead, the button cannot run much too heavy. A charge of 17 g. of litharge gives a button weighing slightly over 15 g.—the usual charge of lead flux (without litharge) for a ½-A.-T. assay is about 30 g.; therefore, if the litharge is to be mixed with the flux in bulk, the proportion of litharge to lead flux is made about 1 to 2, which is equivalent to 10 pounds of litharge in 20 pounds of mixed flux. If a 10-g. crucible is filled about two-thirds full of this mixed flux—which is about the amount commonly used in ½-A.-T. crucible assays—and run, the resulting button of lead will weigh approximately 15 g.—the desired weight. The following flux is practically lead flux No. 2, with litharge added in the above proportion:

No. 3.	Sodium bicarbonate	5 parts
	Potassium carbonate	4 parts
	Borax	2 parts
	Flour	1 part
	Litharge	6 parts
	Salt cover.	

Flux No. 3 is a typical general flux for oxidized ores. For sulphides, the flour may be omitted and from 1 to 4 nails added, according to the amount and nature of the

TABLE V

CRUCIBLE CHARGES FOR GOLD AND SILVER ORES

Ore	Character of Gangue	A. T. Ore	Grams Lead Flux	Grams Soda Bicarb.	Grams Litharge	Grams Potass. Ferrocyanide	Grams Niter	Grams Silica	Grams Argol	No. Nails	Grams Borax Glass	Cover	Remarks
Oxidized	Neutral No. lead	½	30		25							Borax	If cover of salt is used instead of borax, add 3 g. to 5 g. of borax glass.
Quartz	No bases	½			75				2			Borax	Special method. If oxide of iron is present, add soda in proportion.
Quartz	No bases	½	30	30	20							Salt	
Oxidized	Basic. No lead	½	30–40		20			15				Borax	If gangue is oxide or carbonate of iron, add 2 g. or 3 g. of argol.
Oxidized	Basic with barite ($BaSO_4$)	½	40	20	25			15		2		Borax	Borax glass may be substituted for part of the silica.
Galena	Lead, 84 per cent. (Concentrates)	½	20			10						Salt	Heat gradually until mass subsides.

Galena	Silicious	½	15	20	20		5					Salt	Litharge is added, according to the lead contents of the ore.
Lead carbonate	Neutral	½	30	10	15							Borax	Litharge is added, according to the lead contents of the ore.
Iron pyrites	None (Concentrates)	½		35	20		5	15		3		Borax	Collect matte, if any forms, and scorify with lead buttons.
Copper pyrites	Iron pyrites (Concentrates)	½		35	30		5	15		3		Borax	Collect matte, if any forms, and scorify with lead button.
Tellurides \	Silicious	½	30	30	40–80							Salt	If button is hard or brittle, scorify with lead. Scorifier assay preferable.
Tellurides	Silicious	½			80				2			Salt	Special method.
Arsenical		½		15	30	17						Salt	Scorify button. Scorifier assay preferable.
Slags		1	20	40	10						10	Salt	If slag contains matte, add a nail.

sulphides or the proper quantity of desulphurizer calculated. In this flux and similar fluxes, the amount of litharge is kept as low as is consistent with the formation of a lead button of convenient size for cupellation.

55. Another class of fluxes less used employs a large excess of litharge, using it largely as a flux as well as an agent for collecting the precious metals in the charge. Fluxes Nos. 4 and 5 are types of this class.

No. 4.	Sodium bicarbonate	1 part
	Borax glass	1 part
	Litharge	5 parts
	Ore	1 part

To this charge sufficient reducer (or niter, if the ore is itself strongly reducing) is added to bring down a button of convenient size for cupellation, and a cover of salt is put on. (For reducing power of the various reducers, see Art. **27**.) The oxidizing or reducing power of the ore should be determined by a preliminary assay, adding a measured quantity of reducer ($\frac{1}{2}$ g. of charcoal or 1 g. of flour, for example) if the ore is known to be oxidizing, or even if there is any probability of its being oxidizing. The difference between the weight of the button and the weight of lead that would be reduced by the reducer alone represents the oxidizing power of the ore, if the button is lighter than the reducer button; if it is heavier, the difference represents the reducing power of the ore. For this assay, the charge given in Art. **31** may be used, or $\frac{1}{20}$ A. T. of ore may be run down with $1\frac{1}{2}$ or 2 A. T. of flux No. 4, covering the charge thickly with salt.

The great excess of litharge in this flux renders it highly corrosive in its effect on the crucibles, unless a large quantity of silica is added. Another flux of the same class, more commonly used, approaches more nearly the proportions used in fluxes of the first class and largely overcomes this objection. It is made up as follows:

No. 5.	Sodium bicarbonate	3 parts
	Litharge	5 parts
	Borax	2 parts

Reducer or oxidizer, as in No. 4. Salt cover.

56. The accompanying table (Table V) of crucible charges for gold and silver ores, covering both general and special cases, is taken, with a few unimportant changes, from Furman's "Manual of Practical Assaying." The figures in column 4 (grams of lead flux) refer more particularly to lead flux No. 1, but will answer just as well for No. 2 or any similar lead flux. It will be observed that the lead flux forms the base of nearly all the charges. A large excess of litharge is necessary with tellurides in order to oxidize the tellurium, which will otherwise be reduced and make the button brittle.

REFRACTORY-ORE TREATMENT

57. The so-called refractory ores are those which, on account of impurities, mix with the litharge, and prevent the gold from finding its way into the button. Such ores are fluxed with a view of keeping the sulphur, arsenic, antimony, or tellurium out of the lead button. The oxides are also refractory, and must receive special treatment. The reducing power of sulphides and the oxidizing power of oxides have been mentioned under the head of Reducers. The assayer after he becomes accustomed to dealing with such ores does not make preliminary assays, but from their appearance judges the quantity of flux he must add to oxidize the sulphur and balance the reducing power of the oxides.

58. The Common Sulphides.—The common sulphides, which are associated with gold and silver ores and give them reducing powers, are:

Pyrite, FeS_2	Fe, 46.7%;		S, 53.3%
Pyrrhotite, Fe_7S_8 . .	Fe, 60.5%;		S, 39.5%
Arsenopyrite, $FeAsS$.	Fe, 34.4%;	As, 46%;	S, 19.6%
Chalcopyrite, $CuFeS_2$	Cu, 34.6%;	Fe, 30.5%;	S, 34.9%
Chalcocite, Cu_2S . .	Cu, 79.8%;		S, 20.2%
Stibnite, Sb_2S_3 . . .	Sb, 71.8%;		S, 28.2%
Galena, PbS	Pb, 86.6%;		S, 13.4%
Sphalerite, ZnS . .	Zn, 67%;		S, 33%
Millerite, NiS . . .	Ni, 64.4%;		S, 35.6%
Cobaltite, $CoAsS$ . .	Co, 35.5%;	As, 45.2%;	S, 19.3%
Linnæite, Co_3S_4 . .	Co, 58%;		S, 42%

The methods adopted for detecting these sulphides—as well as the arsenides, tellurides, and antimonides that are likewise reducers—may be found in *Blowpiping* and *Mineralogy*.

59. Reducing Power of Sulphides.—The quantity of lead that sulphides, arsenides, etc. will reduce when associated with other minerals in ore can only be determined by preliminary assay. The reducing power of the following pure minerals was determined by litharge and soda:

1 g. of pyrite reduced 12.25 g. of lead.
1 g. of pyrrhotite reduced 8.71 g. of lead.
1 g. of stibnite reduced 7.17 g. of lead.
1 g. of chalcocite reduced 4.38 g. of lead.
1 g. of sphalerite reduced 8.16 g. of lead.

The following equation expresses the reaction that takes place in connection with pyrite:

$$\underset{119.14}{FeS_2} + \underset{1,548.61}{7PbO} = \underset{71.38}{FeO} + \underset{158.92}{2SO_2} + \underset{1,437.45}{7Pb}$$

From this equation, 1 g. of FeS_2 would reduce 13 g. of lead. The theoretical result cannot be obtained unless sodium carbonate is mixed with the charge to induce the formation of a sodium sulphate; the reaction that then takes place is represented by the equation:

$$2SO_2 + 4NaHCO_3 = 2Na_2SO_4 + 2H_2O + 4CO_2$$

The theoretical figures may be approximated closely by using the following charge with pure pyrites:

Pyrite	3 g.
Sodium bicarbonate	9 g.
Litharge	90 g.

While bicarbonate of soda influences the quantity of lead reduced by the sulphides present, it has no influence on carbonaceous reducing agents. When silica is present in a sulphide ore in quantities that will form trisilicates, no lead is reduced, and little lead is reduced when the silica is sufficient to form a monosilicate or a bisilicate.

60. Oxidation of Sulphides.—When an ore contains more sulphides, arsenides, etc. than are needed to furnish the required size of lead button (20 g.), an oxidizing agent

is required to remove this excess. This may be accomplished in three ways:

1. By the addition of potassium nitrate, KNO_3.
2. By roasting the ores so as to remove the sulphur by oxidizing it with the oxygen of the air.
3. By the use of iron nails.

61. Oxidation by Niter.—According to the equation

$$\underset{821.40}{4Pb} + \underset{200.84}{2KNO_3} = \underset{884.92}{4PbO} + \underset{93.58}{K_2O} + \underset{29.81}{NO} + \underset{13.93}{N}$$

1 g. of potassium nitrate oxidizes 4 g. of lead. The niter reacts with the reducing agent before the reducing agent acts with the litharge, so that the reaction between the two is probably as illustrated by the equation:

$$\underset{602.52}{6KNO_3} + \underset{238.28}{2FeS_2} = \underset{158.64}{Fe_2O_3} + \underset{79.46}{SO_3} + \underset{519.12}{3K_2SO_4} + \underset{83.58}{6N}$$

From this equation, 1 g. of niter oxides $\frac{238.28}{602.52} = .39$ g. of pyrite; but 1 g. of pyrite in a soda-litharge charge reduces 13 g. of lead, therefore, 1 g. of niter will oxidize $13 \times .39 = 5.07$ g. of lead.

In an assay charge containing silica, the formation of a lead silicate occurs; but when ferrous oxide is produced the lead silicate is reduced, as expressed by the equation:

$$PbSiO_3 + 4PbO + FeS_2 = FeOSiO_2 + 2SO_2 + 5Pb$$

In this case, 1 g. of pyrite reduces 8.6 g. of lead, and therefore 1 g. of niter should oxidize $\frac{13}{8.6} \times 5.07 = 7.5$ g. of lead.

Example.—From a preliminary assay it has been found that the reducing power of the ore is 5 g. of lead per gram of ore. How many grams of niter must be added to an assay ton (30 g.) of ore to leave only sufficient sulphide to reduce a 20-g. button?

Solution 1.—Soda-litharge charge: $5 \times 30 = 150$ g. of lead, and $150 - 20 = 130$ g. lead to be oxidized; therefore, $\frac{130}{5.07} = 25.6$ g. of niter. Ans.

Solution 2.—Soda-litharge-silica charge: $5 \times 30 = 150$ g. of lead and $150 - 20 = 130$ g. of lead to be oxidized; since 1 g. of niter

should oxidize 7.5 g. of lead, in this charge $\frac{130}{7.5} = 17.3$ g. of niter is required. Ans.

62. Oxidation by Roasting.—The object of roasting ores is to drive off sulphur, arsenic, antimony, tellurium, selenium, etc. For this purpose, the ore is finely pulverized and placed in a roasting dish. The roasting dish is placed in a muffle and the heat gradually raised. The ore should be stirred with a stout iron wire, bent at one end and flattened. When, on stirring, no more burning is seen, the temperature may be raised to a dull red heat. The ore is now a basic ferric oxide, and must be fluxed with suitable reagents.

In roasting, the heat must not be too severe at first, in order to prevent the ore louping or becoming fused; for, if this occurs, a complete oxidizing roast cannot take place.

If the ore is placed where the temperature is too hot, and there is a current of air, the volatilization will occur so rapidly that fine gold will be wafted out of the dish by the fumes, particularly if arsenic, antimony, or tellurium is present.

If all the impurities are not removed during roasting operations, they may have a reducing power; moreover, they may carry gold and silver with them into the slag.

63. Desulphurization With Nails.—When ore contains a small proportion of sulphur and sufficient silica to form a subsilicate, it may be desulphurized by the addition of several twenty-penny nails to the charge. The iron forms an iron sulphide, and if more iron is dissolved than the slag requires a matte is formed.

In using nails to decompose sulphide ores, the button can often be rendered soft and the matte gotten rid of by removing the nails about 10 minutes before the crucibles are taken from the furnace, and then raising the heat so as to render the slag thoroughly fusible and as far as possible to decompose the matte at the expense of the oxides in the slag. If but a small amount of matte is formed, this method will usually decompose it and carry the gold and

silver all into the lead. In some cases, it is necessary to add an excess of litharge to the flux. The litharge will first pass into the slag, and later, during the decomposition of the matte, the lead will pass into the button and the matte become oxidized and its iron and copper constituents pass into slag. If care is taken in proportioning the charge and too great an amount of reducer is not added, it is usually possible to obtain a lead button of about the desired weight; for the sulphur will first pass into the matte and subsequently act as a reducer on a portion of the litharge in the charge, thus bringing the button to about the right size.

When matte forms, the lead button is separated from it with great care; for, if the matte is brittle, a little of it is almost certain to fly off, and become lost, and if the ore is rich this will give low results. However, when the matte is separated from the button, it is pulverized with the slag, mixed with 30 g. of litharge, 1 g. of flour, more twenty-penny nails, sufficient silica to form a silicate, and covered with borax glass. The button obtained is cupeled with the first one, or, if the combined buttons are too large, they are scorified to the proper size before cupeling.

If impurities other than sulphur are present, they will go into the matte, in which case the matte and button should be scorified together without separation. In the scorification assay, the undesirable elements are oxidized and volatilized, thus passing off as fumes. The scorification method, however, gives slightly higher results for silver.

Another method sometimes followed is to pulverize the matte and roast it dead, then place the residue in a scorifier with the first button obtained, add silica test lead, cover with borax glass, and scorify.

If silica is not added to the matte or to the roasted ore, it will attack the crucible, and the slag will be pasty.

ASSAYING

(PART 3)

SMELTING ASSAYS

CRUCIBLE ASSAYING

1. **Size of Assay Charge.**—The most common practice among assayers in the United States is to use ½ A. T. of ore, a 10-g. crucible, and run duplicate charges; this method affords a check on the accuracy of the work. If the gold in the ore is insufficient to furnish a bead that can be weighed readily and accurately, two or more ½-A.-T. charges are run and the lead buttons obtained are cupeled together. If the operator desires a check it will probably be better to use 1 A.-T. ore and a 20-g. crucible, and run duplicate assays, rather than a number of ½-A.-T. fusions. Assay charges larger than 1 A. T. need not be run in duplicate, on account of the quantity of ore taken for assay. In the I. C. S. assay office, 15-g. crucibles are usually taken for ½-A.-T. ore charges to guard against any possibility of boiling over. The 10-g. crucible is filled about two-thirds full with such a charge, while the 15-g. crucible is but a little more than one-half full. In no case should a crucible be more than three-fourths full.

2. **Determining the Flux.**—The flux used is to a considerable extent dependent on the composition of the ore. If the sample is in lump form, the predominating element can be determined by simple inspection or by the blowpiping or wet tests for minerals. If the sample is pulverized or

amorphous, so that the inspection and physical tests are not readily made, a little of it is panned and examined with a magnifying glass, then blowpipe and wet tests are to be made. The nature of the ore having been determined, the table of ore charges given in *Assaying*, Part 2, is consulted for a suitable flux. If the general composition of the ore cannot be ascertained by examination, one of the stock fluxes is tried; if it does not prove satisfactory, it will at least indicate where the difficulty lies so that it may be corrected. It may be necessary to calculate a flux as explained; however, in order to do this, if the analysis of the gangue rock is not given in the table of rock analysis, it must be ascertained by quantitative analysis. Such drastic measures, however, are seldom necessary: if they were, the cost of assaying would be very much higher than it is.

3. **Mixing the Charge.**—Whatever method of mixing a crucible charge is adopted, the mixing must be thorough, so that the lead as it is reduced can come in contact with the gold freed by pulverization and fusion. While there is very much more lead than gold in the ore, yet it is evident that if the mixing is not thorough the two may escape each other. To mix a charge, some assayers put the flux and litharge on an oil or rubber cloth and the weighed ore charge on top of them. The materials can be thoroughly mixed by lifting first one corner and then another so as to roll the charge over and over, as described when explaining the mixing of samples in *Assaying*, Part 1. When thoroughly mixed, the material is placed in the crucible, and the cover, with nails, if necessary, is added. Another method is to put the flux and litharge in a crucible, and the weighed ore on top of it. The materials are thoroughly stirred with a spatula to mix them, after which nails are inserted point down, if the ore is a sulphide, and then the cover of salt or borax glass added.

4. **Fusing the Charge.**—If the gold in the ore is all coarse, rapid fusion is not so objectionable as where the ore is fine and incased or attached to gangue. With some ores, the gold is so fine that it is never set free entirely within

the limits of fine-crushing 80- to 200-mesh screens. If the lead settles through the charge before the gangue is sufficiently decomposed to liberate the gold, the latter will remain in the slag. By a comparatively slow fusion at the commencement of the heat it is possible to delay the settling of the lead through the charge until the flux has commenced to decompose the gangue. When the muffle is used, it should be at a red heat before the crucibles are inserted, and the door is to be kept closed except when it is deemed necessary to open it for inspection. The charge swells when first heated and it is to be closely watched during the first 15 or 20 minutes, to see that it does not boil over. If too much borax is mixed with oxidized ore, the charge will swell and boil over; unfused borax is particularly bad in this respect on account of the water it contains. Too much soda will cause boiling, and rapidly heating a charge of niter is almost sure to cause frothing over. If the crucible commences to froth up, the boiling over can be prevented by checking the heat quickly. Sometimes, boiling over can be prevented by opening the muffle door and keeping it open until the muffle has cooled down; or a teaspoonful of common salt thrown into the crucible will serve the same purpose; in either case, the remedy is to be applied quickly. The result of an assay that has run over is not to be accepted.

As soon as the fusion becomes perfectly quiet and no more *cooking* is heard, the fusion is completed. Leave the crucibles in the muffle a few minutes longer, and finish quite hot, to make the slag perfectly fluid. Then, remove, and take out the remains of the nails, if any were used, with the small crucible tongs, washing them in the slag and tapping them lightly against the side of the crucible as they are withdrawn, to detach any adhering globules of lead. Next, give the crucible a slight whirling motion by moving the hand around in a horizontal circle; tap it gently on the ledge of the furnace, to collect and sink any fine globules of lead that may be held in suspension by the slag; and then pour the contents into its proper hole in the warm mold.

PRACTICAL SUGGESTIONS

5. Office System.—If an assay office is not conducted according to a rigid system, endless trouble is certain to result. From the time a sample is received until the assay certificate is made out, it must be systematically handled and cared for to prevent its being confused with another sample. If this is not done, the assayer can never be absolutely sure of his work.

6. Marking Samples.—The pulverized samples should be put into envelopes, bottles, or boxes, marked with the assayer's number of the sample or lot, the name of the sender or mine, the metals to be determined, and any additional remarks that may be considered necessary. It is a good practice also to put the date of receiving the sample on the envelope or bottle. Envelopes are much more convenient for samples than boxes or bottles, and are cheaper and less bulky. Special sample envelopes are made for assay samples, which close tightly, without sealing, in such a manner that none of the pulp can leak out nor any dirt find its way in, and at the same time they can be opened very readily and without damaging the envelope. Large assay offices usually have their name and the blank form for marking the samples printed on their sample envelopes.

Many assayers, instead of marking samples in the manner just stated, put only the name and lot number on the envelope, and then under the lot number in a notebook enter a more detailed description of the sample, its character, etc.

7. Numbering Samples.—Assayers usually employ running numbers for their samples; that is, the samples are numbered consecutively, and the numbers are never repeated. This avoids confusing different samples from the same person or mine.

8. Weighing and Furnace Work.—When weighing samples, a record should be kept in a notebook of the order in which the samples are weighed and the quantity of pulp taken for assay. Number each day's work consecutively,

from 1 upwards. The same order should then be preserved all through fusion, cupellation, and parting, to the final weighing of the gold. This will avoid the confusion that would arise from the assayer losing track of buttons that came from any particular ore. If a single one of a batch of assays goes astray, the entire lot might as well be thrown out, as the assayer can never be positively sure just which assays are out of place, and a result to which the least doubt attaches is worse than useless. In addition to keeping his assays in a fixed order and to help him in so doing, it is well

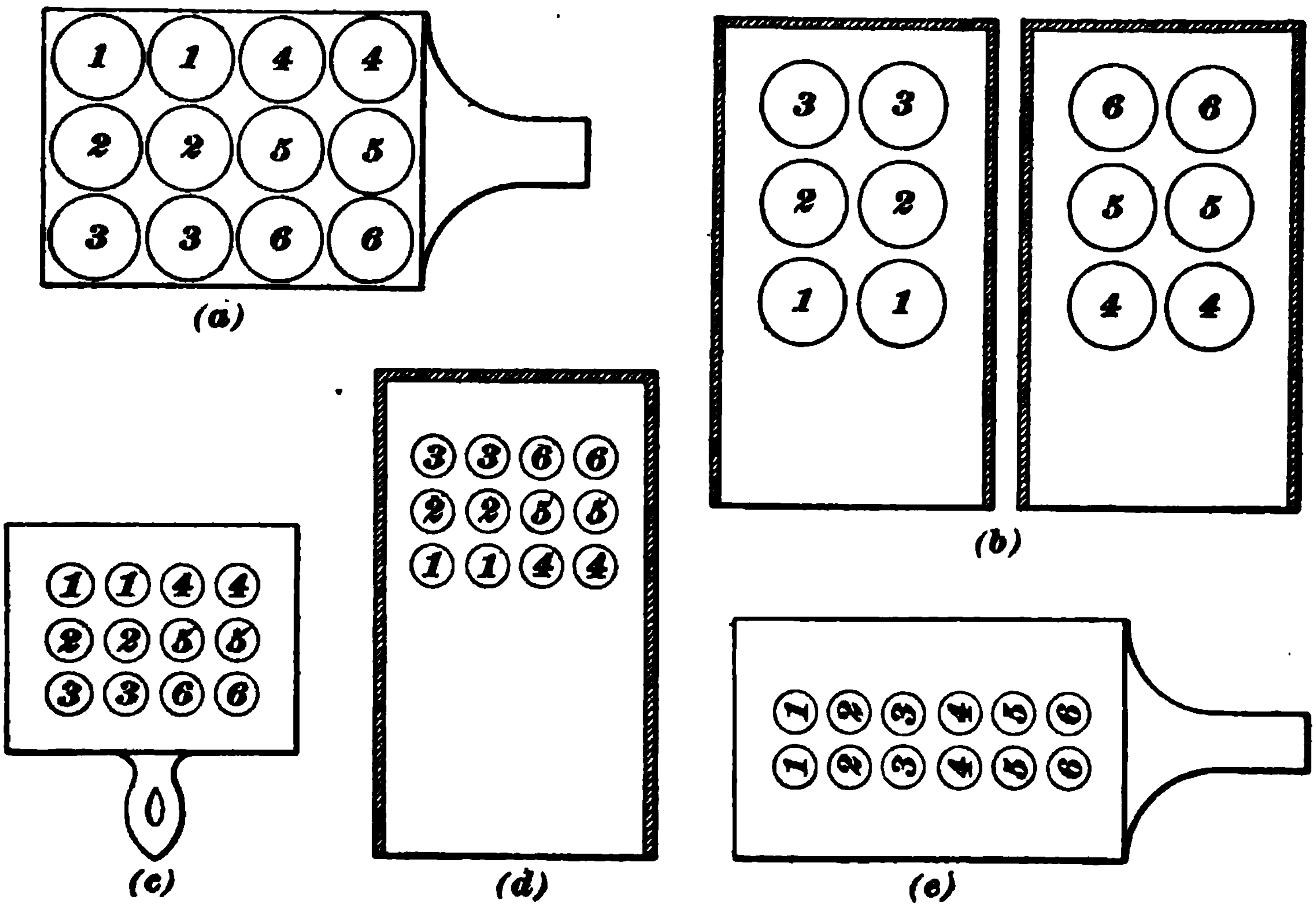


FIG. 1

for the beginner to mark his crucibles and scorifiers plainly, in several places, with the numbers of the assays they contain, using *reddle* (red ocher), in the form of either chalk or paint, as it is not affected by the heat of the furnace. As a further precaution, the numbers of the buttons can also be scratched on the sides of the cupels.

9. Handling Assays.—In Fig. 1 is illustrated a convenient system of handling the assays from the weighing of the charge to the weighing of the gold and silver buttons, in order to avoid confusion. The scheme is here worked out

for six duplicate assays and a two-muffle furnace; but the same principles are applicable to any number of assays in any furnace.

The charges as they are weighed out are placed in their respective crucibles or scorifiers on the cupel board, with No. 1 at the right-hand back corner, and the rest successively, as shown at (*a*). With this arrangement, the assayer never has to reach over already charged crucibles or scorifiers to set assays in their proper places, and thus run the risk of tipping over a crucible or brushing some of the ore off a scorifier charge with his sleeve.

The assays are carried from the weighing room to the furnace on the cupel board. The board is set sidewise on the bench, with Nos. 3 and 6 to the front, as shown at (*a*), and the assays are put into the muffles in the natural order of their positions—those at the front side of the board going to the back of the muffles, and vice versa, as shown at (*b*). When the fusion is finished, Nos. 1 and 4, which are in front in the respective muffles, are naturally withdrawn first, and are poured into the back holes of the mold, as at (*c*), bringing the assays once more into the original order. In the cupels in the furnace, the buttons are again in reversed order, as at (*d*), and the cupels are drawn in their proper sequence, as at (*e*), and carried to the weighing room.

10. To Heat the Muffle.—To raise the heat of the muffle rapidly, build a good fire under it, heaping the coal well up and allowing it to burn down to glowing coals; then, as soon as the muffle commences to show a dull red heat, throw a little charcoal or coal *into the muffle* and shut the door. The coal in the muffle will take fire from the heat of the muffle, and its heat added to that of the fire below will quickly bring the muffle to a good fusing heat, when the coals inside may be withdrawn and the crucibles or scorifiers inserted. A shovelful of burning coals from the grate will accomplish the same purpose in even less time. The same scheme may be employed in case the heat should be allowed to fall low during fusion. The coal or charcoal is placed at

the front of the crucibles and are near the muffle door. In cupeling, this also assists in opening frozen cupels.

The use of coal in the muffle should be avoided, as the ash is liable to form a sticky slag that is troublesome. The use of wood or charcoal is not so objectionable.

11. Accidents.—In spite of the greatest care, an occasional accident is unavoidable in an assay office. Crucibles, scorifiers, or cupels may be spilled, crucibles may boil over, a defective scorifier may be corroded through, or cupels may crack and let the molten button through on to the floor of the muffle. Litharge and oxidizing lead corrode the muffle very rapidly; if the lead is left on the floor of the muffle, it will soon eat its way through, and, once started, the muffle soon goes to pieces. Most slags are not very actively corrosive, but slag is a great nuisance on the floor of the muffle, even in very small quantities, as it causes the dishes to stick, and if they are not released very cautiously, a spill is liable to result.

The standard remedy for all such troubles in the muffle is bone ash. If a vessel boils over or spills, remove it at once and throw in a handful of bone ash. This will mix with the slag and form a thick paste, which can be readily removed with a scraper. If any lead is left on the floor of the muffle, throw in some more bone ash on top of it. The bone ash will absorb the litharge as it forms, and save the muffle to a considerable extent. It is advisable to keep the floor of the muffle always thinly covered with bone ash; this will afford considerable immediate protection in case of spills, etc., and will also prevent dishes sticking to old slag spots on the bottom.

CALCULATING ASSAYS

12. The calculation of the results of gold and silver assays is a matter of simple arithmetic. The rule for the calculation of the number of ounces of precious metals per ton of ore must be kept in mind; the rest of the calculation is mere multiplication. The rule is repeated here in the shape of a formula:

$$\frac{\text{weight of button in mg.}}{\text{weight of ore taken in A. T.}} = \text{number of oz. per ton}$$

For example, if the button from an A. T. of ore weighs 213 mg. before parting and the gold from parting weighs 13 mg., the contents of the ore in gold and silver are figured as follows: 213 mg. (gold and silver) − 13 mg. (gold) = 200 mg., the weight of silver in the button.*

Then, since there are 200 mg. of silver from a 1-A.-T. charge of ore, this value in the formula, gives $\frac{200}{1} = 200$ ounces silver per ton, and the gold (13 mg.), when transposed to the formula gives $\frac{13}{1} = 13$ ounces gold per ton.

If $\frac{1}{2}$ A. T. of this ore is taken, the buttons will weigh half as much as the buttons from 1 A. T.; but as the ore charge also is only half as large as in the 1-A.-T. assay, the weight of the buttons divided by the weight of ore used gives the same result as in the 1-A.-T. calculations; thus,

$$\frac{100}{\frac{1}{2}} = 100 \times 2 = 200 \text{ ounces silver per ton,}$$

and $$\frac{6.5}{\frac{1}{2}} = 6.5 \times 2 = 13 \text{ ounces gold per ton}$$

If only $\frac{1}{10}$ A. T. of the ore were taken, the silver and gold would weigh 20 mg. and 1.3 mg., respectively, and the figures would read

$$\frac{20}{\frac{1}{10}} = 20 \times 10 = 200 \text{ ounces silver per ton,}$$

and $$\frac{1.3}{\frac{1}{10}} = 1.3 \times 10 = 13 \text{ ounces gold per ton}$$

The value of pure gold is definitely fixed at $20.67 a troy ounce, and this value is the same in all civilized countries. Most custom assayers, however, figure gold at $20 per ounce, as this value is much more convenient for calculation.

The value of silver fluctuates considerably, and the silver values in an ore are figured at the prevailing market price of silver. In all the calculations in these Sections, gold is figured at $20.67 an ounce and silver at 60 cents an ounce. At these prices, the value of the ore in the preceding examples would be,

*For the purpose of calculating, the gold and silver are considered separately, as if they were in separate buttons, one pure gold and the other pure silver.

Gold: 13 ounces @ \$20.67 per ounce = \$268.71
Silver: 200 ounces @ \$0.60 per ounce = 120.00

Total value of ore in gold and silver, \$388.71 per ton

13. Ores With Metallic Scales.—When an ore contains particles of metallic gold and silver too coarse to pass through the screen with the pulp, these scales must be assayed separately, the amount of gold and silver they contain determined and added to the total gold and silver in the pulp, and the sum divided by the number of assay tons in the entire sample—pulp and scales—to obtain the total gold and silver in each assay ton of the sample.

If the sample is known to contain metallic scales, it may be weighed before crushing. After the sample is pulverized, the pulp and scales are weighed separately. Their combined weight should be only a trifle less than that of the original sample, if the work has been carefully done. If the scales do not show in the lump sample, it would naturally be crushed without weighing, and the combined weight of the pulp and scales would then have to be taken as the weight of the original sample; hence, care should be taken, in bucking samples, to lose as little of the sample as possible. If the bucking is carefully done, the loss of ore in an ordinary-sized sample will be so small that the error in the calculations from this cause will not be appreciable.

14. If there is any considerable quantity of scales, they should be scorified in the usual manner and the lead button cupeled. If there is only a small quantity, they may be wrapped in lead foil and cupeled directly. The gold-silver button is weighed and parted us usual and the gold weighed. The weight of the gold subtracted from the weight of the button will give the weight of silver in the scales from the entire original sample.

15. The pulp is assayed in the usual manner by either the scorification or the crucible process, using the regular charge—$\frac{1}{10}$, $\frac{1}{2}$, or 1 A. T. The results may then be calculated as follows:

Let A = weight of the pulp, in grams;
B = weight of the scales, in grams;
C = assay value of pulp in ounces of gold or silver per ton (or mg. per A. T.);
D = weight of the gold or silver in the scales, in milligrams.

Now, $\frac{A}{29.166}$ = weight of the pulp in assay tons, and $\frac{A+B}{29.166}$ = total weight of sample in assay tons. $\frac{AC}{29.166}$ = number of milligrams of gold or silver in the pulp; and if the weight of gold or silver in the scales be added to this, then,

$$\frac{AC}{29.166} + D = \begin{cases}\text{total number of milligrams of} \\ \text{gold or silver in entire sample}\end{cases}$$

To obtain the weight of gold or silver in 1 A. T., divide the total weight of gold or silver in the sample by the weight of the sample in assay tons; thus,

$$\frac{\frac{AC}{29.166} + D}{\frac{A+B}{29.166}} = \frac{AC + 29.166\,D}{A+B} = \begin{cases}\text{milligrams of gold or silver} \\ \text{per A. T. (or oz. per ton)}\end{cases}$$

EXAMPLE.—Suppose that the pulp from a sample weighs 107.8 g. and the scales weigh .235 g. (235 mg.). If the scales contain 135 mg. of gold and 60.5 mg. of silver, and the pulp assays 3.28 ounces of gold and 1.7 ounces of silver per ton, what is the total gold and silver contents of the ore, in ounces, per ton?

SOLUTION.—The complete solution is as follows:
A = 107.8; B = .235; C = 3.28 for gold and 1.7 for silver; and D = 135 for gold and 60.5 for silver. Then,

$$\frac{A}{29.166} = \frac{107.8}{29.166} = 3.696 \text{ A. T., weight of pulp,}$$

and

$$\frac{A+B}{29.166} = \frac{107.8 + .235}{29.166} = \frac{108.03}{29.166} = 3.704 \text{ A. T., total weight of sample.}$$

The total weight of gold or silver in the pulp equals

$$\frac{AC}{29.166} = \frac{A}{29.166} \times C = 3.696\ C;$$

hence, for gold,

$$\frac{AC}{29.166} = 3.696 \times 3.28 = 12.123 \text{ mg. gold in pulp,}$$

and for silver,

$$\frac{AC}{29.166} = 3.696 \times 1.7 = 6.283 \text{ mg. silver in pulp}$$

The total weight of gold or silver in the entire sample = the weight in the pulp + the weight in the scales, or $\frac{AC}{29.166} + D$; hence, the total gold in the sample = 12.123 + 135 = 147.123 mg., and the total silver in the sample = 6.283 + 60.5 = 66.783 mg.

Then, since the entire sample weighs 3.704 A. T.,

$$\frac{147.123}{3.704} = 39.72 \text{ mg., total gold in 1 A. T. of ore (or 39.72 oz. per ton)};$$

and

$$\frac{66.783}{3.704} = 18.03 \text{ mg., total silver in 1 A. T. of ore (or 18.03 oz. per ton)}$$

The preceding calculations are given in full merely to illustrate more fully their principle. The same result may be obtained with much less work by simply substituting the values of A, B, C, and D in the final formula $\frac{AC + 29.166\,D}{A + B}$ = mg. per A. T., or ounces per ton; thus, for gold,

$$\frac{107.8 \times 3.28 + 29.166 \times 135}{107.8 + 0.23} = \frac{353.6 + 3{,}937.4}{108.03} = \frac{4{,}291}{108.03}$$

$$= 39.72 \text{ mg. gold per A. T. (or ounce per ton)};$$

and for silver,

$$\frac{107.8 \times 1.7 + 29.166 \times 60.5}{108.03} = \frac{183.3 + 1{,}764.5}{108.03} = \frac{1{,}947.8}{108.03}$$

$$= 18.03 \text{ mg. silver per A. T. (or ounce per ton)}.$$

16. Horn silver (chloride of silver) in an ore may cause the same trouble as metallic gold or silver, as it is malleable and flattens out into scales instead of breaking up and passing through the screen with the pulp. These scales are treated just like metallic scales, and the results are calculated in the same manner.

If there is only a very small quantity of scales and the scales are not very coarse, they may be ground down fine enough to pass the screen by placing them on the bucking board, covering them with a little of the pulp that has already passed the screen, and grinding heavily for a few minutes. Then sift, and if any scales are still left, grind again in the

same way, and repeat until they all pass through the screen. Mix the sample with unusual care when weighing out the charges, to make sure that the scales are evenly distributed. This involves a little more work in the preparation of the sample, but saves the trouble of the extra assay and the calculations necessary when the scales are assayed separately.

EXAMPLES FOR PRACTICE

1. *Scorification Assay.*—Ore charges, $\frac{1}{10}$ A. T. The gold-silver buttons weigh 1.72 and 1.76 mg., respectively, and the gold from parting weighs 0.98 mg. (*a*) How many ounces of gold and silver does the ore contain per ton? (*b*) What is the value of the ore per ton?

Ans. (*a*) Gold, 4.9 oz. per ton; Silver, 12.5 oz. per ton
(*b*) Gold, $101.28 per ton; Silver, 7.50 per ton; Total, $108.78 per ton

2. *Crucible Assay.*—Ore charges, $\frac{1}{2}$ A. T. Weight of gold-silver buttons, 2.96 and 2.90 mg., respectively. Weight of gold, 1.03 mg. (*a*) How many ounces of gold and silver does the ore contain per ton? (*b*) What is the value of the ore per ton?

Ans. (*a*) Gold, 1.03 oz. per ton; Silver, 4.83 oz. per ton
(*b*) Gold, $21.29 per ton; Silver, 2.90 per ton; Total, $24.19 per ton

3. *Crucible Assay.*—Ore charges, 1 A. T. Weight of gold-silver buttons, 0.78 and 0.81 mg., respectively. Weight of gold, 1.34 mg. (Buttons would have to be inquarted.) (*a*) How many ounces of gold and silver does the ore contain per ton? (*b*) What is the value of the ore per ton?

Ans. (*a*) Gold, 0.670 oz. per ton; Silver, 0.125 oz. per ton
(*b*) Gold, $13.85 per ton; Silver, 0.07 per ton; Total, $13.92 per ton

NOTE.—When the quantity of silver contained in an ore is as small as this, it is usually ignored entirely when calculating the value of the ore and reported as a "trace." Such small quantities of silver are never considered in buying and selling ores, and in such a case as the above, only the gold would be paid for. In assaying gold ores known to contain so little silver that it may be safely neglected, it is a common practice to add enough silver to the assay charges—in the shape of silver foil or a small crystal of nitrate of silver—to inquart the gold buttons. This silver will go into the lead buttons along with the gold and silver in the ore, and when they are cupeled the assayer has his buttons all ready to part, and does not have to take the risk of losing them in inquarting with the blowpipe, to say nothing of the work saved. When only the gold is determined, it is not necessary to weigh the buttons before parting, except as a check on the assaying.

4. *Ore Containing Metallic Scales.* Weight of pulp, 138.67 g. Weight of scales, 1.23 g. Scales contain 832.4 mg. silver and 8.58 mg. gold. The pulp is assayed by the crucible process, using $\frac{1}{2}$-A.-T. charges of ore; the gold-silver buttons obtained weigh 107.54 and 107.28 mg., respectively, and the gold from parting weighs 2.26 mg. (*a*) How many ounces of gold and silver does the ore contain per ton? (*b*) What is the value of the ore per ton?

$$\text{Ans.}\begin{cases} (a)\begin{cases}\text{Gold,} & 4.03 \text{ oz. per ton}\\ \text{Silver,} & 384.20 \text{ oz. per ton}\end{cases}\\ (b)\begin{cases}\text{Gold,} & \$\ 83.30 \text{ per ton}\\ \text{Silver,} & \underline{230.52} \text{ per ton}\\ \text{Total,} & \$313.82 \text{ per ton}\end{cases}\end{cases}$$

SCORIFICATION ASSAY

17. Explanation of Scorification.—Scorification includes a combination of fusion, roasting, sublimation, and oxidation, particularly oxidation. The ore is mixed with granulated lead and floats on a lead bath, where sulphur, arsenic, and tellurium are removed in part by volatilization and oxidation. The lead begins to oxidize and carries with it as a slag the gangue and oxides of the non-volatile metals. At this point, a little powdered charcoal in tissue paper is placed on the fusion by means of tongs, and this reduces some of the lead, which in falling through the slag collects the gold and silver.

The scorification assay is made in a muffle, which should have a temperature of at least 1,000° C. The scorifiers are placed in a hot place on the furnace previous to their being charged into the muffle, the object being to heat them so that the intense heat of the muffle will not crack the dishes.

18. Scorification Charge.—The ordinary ore charge for the scorification assay is $\frac{1}{10}$ A. T. in $2\frac{1}{4}$- or $2\frac{1}{2}$-inch scorifiers. Occasionally $\frac{2}{10}$ A. T. is used. The principal fluxes are test lead and borax glass. Litharge, and occasionally soda and niter, are used in special cases as covers. Silica added to the charge for a basic ore will save the scorifier. The scorifier charges for various ores, recommended by Furman, are given in Table I. The charges are figured on a basis of a $\frac{1}{10}$ A. T. of ore.

TABLE I

SCORIFIER CHARGES

Ore, $\frac{1}{10}$ A. T.	Grams of Test Lead	Grams of Borax Glass	Remarks
Galena	15–18	Up to 0.5	
Galena, with zinc blende and pyrite	20–35	0.4–0.8	
Iron pyrites	30–45	0.3–0.8	
Arsenical pyrites	45–50	0.3–1.5	High temperature. Addition of litharge helps assay.
Gray copper	35–48	0.3–0.5	Low temperature.
Zinc blende	30–45	0.3–0.6	High temperature. Addition of oxide of iron helps assay.
Copper ores and mattes	35–40	0.3–0.5	Low temperature. If necessary, the button should be rescorified with lead.
Tellurides	50	0.3	Add cover of litharge, and rescorify the button.
Silicious	25–30		
Basic	25–30	0.5–2.0	If the ore contains much lime or magnesia, the addition of sodium carbonate helps assay.
Basic with barium sulphate	25–30	0.5–1.5	Addition of sodium carbonate helps assay.
Lead carbonate	10–15	Up to 0.5	

About half of the test lead is put in the scorifier. (The lead need not be weighed, but may be measured with sufficient accuracy by a shot measure or small crucible, the capacity of which is known.) The ore charge is then weighed out and brushed in on top of this, and the two are thoroughly mixed with a small spatula. The remainder of the lead is now put on as a cover, and the borax glass on top of this. The borax may be added by measure or by pinches, a little practice enabling the assayer to guess sufficiently close to the correct weight; however, if too much borax is used, the slag will cover the bath of metal too soon. Duplicate charges are always run, to prevent the possibility of errors from carelessness or accident going undetected. If the ore is low grade, a number of assays should be made, the resulting buttons placed in a scorifier, together with a little borax glass, and then scorified to the proper size for cupellation. If the two buttons do not check very closely, the assay should be repeated.

19. Charging the Muffle.—The scorifiers are charged into the hot muffle, and the door is closed and kept closed until the charge melts down and active scorification commences, when it is opened to admit a plentiful supply of air. If the muffle is not hot enough, when the scorifiers are charged in, gold may remain in the slag. The fused charge now displays a clean, mirror-like surface of glowing, molten lead, with a narrow ring of slag around the sides of the scorifier. This slag is formed by the fusion of the gangue of the ore with the borax and the litharge formed by the oxidation of the melted lead in the current of air flowing through the muffle.

20. Oxidation of the Charge.—After the door is opened, oxidation goes on more rapidly than before. Sulphur, arsenic, etc. are oxidized by the action of the litharge and of the air-current, and the fumes pass off through the opening in the rear of the muffle. Any metallic gold or silver in the ore is immediately dissolved by the lead as it sinks through the charge. Compounds of gold or silver with other

elements—sulphides, tellurides, etc.—are broken up by the action of the litharge, the gold or silver and the lead being reduced to metals and sinking together into the button, while the oxygen of the litharge converts the sulphur, etc. into gaseous oxides, which pass off through the back of the muffle.

As the oxidation progresses, the slag ring gradually spreads inwards toward the center, until it finally closes over the top of the lead button, preventing further oxidation. This marks the end of the scorification.

21. Pouring the Melt.—The door is now closed for a few minutes and the heat raised, to make the slag thoroughly liquid; the scorifier is then removed, tapped lightly on the ledge of the furnace, to settle any suspended shots of lead, and the contents poured into a mold. The mold should be warmed beforehand, or the sudden chilling may cause the slag to break up before the lead has solidified and thus spatter the button; some of the lead, too, is apt to chill in small shots instead of going into the button. The slag should be clean, liquid, and glassy, and the lead should all be collected in one button at the bottom and not scattered in shots through the slag. This button should be soft and malleable.

As soon as the assay is cool, the button may be separated from the slag by a few blows with a hammer and then beaten into the form of a cube, to make it easy to handle with the cupel tongs. The buttons are now ready for cupellation, provided they are not too large. The cubes should be about ½ inch on a side—this size button will weigh about 15 g. If much larger than this, or brittle, they should be rescorified with borax glass and a little more lead, if necessary, and this should be repeated until they are of the proper size and purity. The loss of precious metals is less by this method than where a large button is cupeled directly, as the only loss in scorification worth mentioning is through volatilization, and is very small, while in cupellation the principal loss is through silver being carried into the cupel by the litharge, in addition to which there is a loss from volatilization nearly or quite as large as that in scorification.

22. Scorification Slags.—Most metal oxides furnish slags that leave a distinctive color on the scorifying dish: *Copper* gives a light green slag. *Copper and iron* give a greenish brown, but not quite a pinchbeck brown. *Iron* leaves a brown enamel on the scorifier. *Iron and copper* furnish a brownish-green slag. *Manganese* leaves a yellow glaze on the dish. *Cobalt* gives a dark blue enamel, which will vary in color according to the quantity of oxide in the ore. *Chromium* furnishes a reddish-brown slag of peculiarly rich tone. *Lead* gives a lemon-colored enamel to the scorifying dish. *Nickel* furnishes a chocolate-brown enamel.

CUPELLATION

23. Composition of Cupels.—Cupels are used to absorb the litharge that forms when the lead in a button containing gold and silver is oxidized in the muffle. Several porous substances, such as leached wood ashes and lime and magnesia, have been tried for cupels, but none has given the general satisfaction of bone ash.

The bones for cupels are calcined and then washed with an aqueous solution of ammonium chloride, NH_4Cl. This reacts with the calcium carbonate and lime in the bone and forms calcium chloride, $CaCl_2$, which is removed by water. Carbon dioxide, carbon monoxide, and nitrous fumes decompose below the temperature of cupellation, and if present in a cupel will cause "spitting" due to little explosions.

In assay offices it is customary to purchase cupels, and it is only in small assay offices and places where supplies cannot readily be obtained that they are made by hand. Those made by machine presses are more uniform and dense than those made by hand. Cupels must be dried gradually and not hurriedly, for if dried too fast they will crack, and if not thoroughly dried they will crack when placed in the muffle.

The bone ash is wetted with 10 per cent. of water, to which is added a little molasses, or stale beer, or potassium carbonate, K_2CO_3. After a batch of cupels has been made and thoroughly dried, the cupels should be tested to ascertain

the quantity of silver they will absorb. For this purpose .2 g. of pure silver should be mixed with 20 g. of lead in a cupel and placed in the muffle. The loss in silver should not exceed 2 per cent., and is determined by weighing the silver before and after cupellation.

24. Cupel Absorption.—A good cupel will efficiently absorb just about its own weight of litharge. As a considerable part of the litharge from the oxidation of the lead buttons passes off in fumes, a cupel may be safely used for a button somewhat heavier than itself. It is much better, however, to have the cupel about a quarter or a half heavier than the button, as a cupel nearly saturated with litharge absorbs any additional litharge more and more slowly, and absorption may cease entirely—even with some untouched bone ash at the bottom of the cupel—just at the time when absorption should, if anything, be most rapid. The bowl of the cupel will usually hold more lead than the cupel will efficiently absorb. If a large button is being cupeled, and it is seen that the cupel will have difficulty in absorbing all the litharge, a second hot cupel may be inverted, and the cupel containing the button set on this; cupellation will then proceed, though somewhat slowly, the excess of litharge passing into the lower cupel. It is much better, however, to reduce lead buttons to the proper size by scorification and not take any risks.

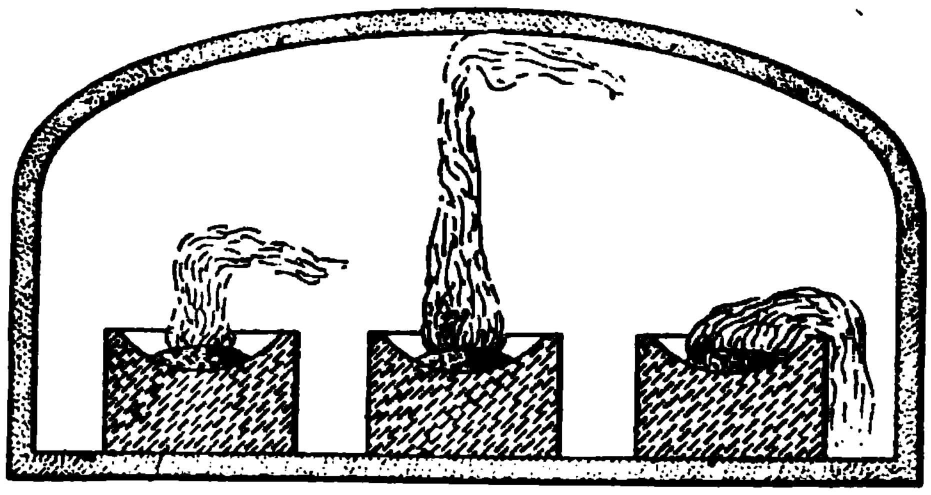
25. Cupellation.—When the muffle has been raised to a red heat, the empty cupels are placed in the front half of the muffle and allowed to remain in this position until they have attained the heat of the muffle. This is also a precautionary measure against spitting, for if the cupels have not been thoroughly dried or if they have absorbed gases, the lead button which melts at 326° C. would spit when the gases were expelled at a temperature between 500° C. and 700° C. When the cupels have reached the temperature of the muffle, the buttons are placed in their proper cupels with cupel tongs. If the door of the muffle is now closed the buttons will melt rapidly.

The surface of the melted lead is at first covered with a gray-black scum, composed of slag and oxide; this, however, will disappear, provided the button was comparatively free from metalloids and the difficultly fusible base metals. When the lead reaches a temperature 675° C. the scum should disappear and a glowing surface of lead appear. This is called *uncovering*, or *opening up*, the lead button, and when it occurs the lead is seen to be in active movement, or is said to *drive*, owing to the rapid oxidation that has commenced.

Little flakes of litharge form and slide down the convex surface of the molten lead to the cupel, which being porous immediately absorbs them. In case the temperature is not too low, crystals of litharge termed *feathers* will form on the side of the cupel toward the muffle door. If the temperature is too low the feathers form low down in the cupel, and indicate that the cupel is not sufficiently hot to absorb all the litharge; when the temperature is about right the feathers will form near the upper rim of the cupel. If the draft is strong through the muffle, feathers will form, even if the temperature is somewhat above 700° C. When the percentage of lead in the fusion is materially decreased, the litharge is thrown off from the button in larger specks, and the molten metal assumes a more rounded form. When the last of the lead goes off, the silver-gold button takes a brilliant film of colors and the button seems to revolve axially. The colors then disappear, the bead becomes dull and then again takes a silvery tinge.

If the temperature of the muffle is below the melting point of silver (961° C.) or the melting point of gold-silver alloy (750° C.), or if the cupel is withdrawn from the furnace, the bead becomes suddenly very bright at the moment of solidification, owing to the release of the latent heat of fusion, which raises the temperature very much for a short time. When the brightening occurs, the bead is said to **blick,** or **flash.** Silver and gold beads containing small amounts of lead, copper, platinum, and palladium do not brighten so noticeably as pure gold or silver beads, and beads that contain rhodium, iridium, ruthenium, osmium, or osmiridium do not brighten at all.

26. Temperature and Cupellation.—The heat of the muffle during the cupellation of the lead button from high-grade silver ores is of great importance, as silver is somewhat volatile, and the loss from volatilization is greatly increased by cupeling at a high heat. With gold ores, the heat is less important, as the loss of gold from volatilization is practically nothing. As a general rule, however, *do not carry the temperature of the muffle beyond a good, strong, red heat while cupeling, and for ores rich in silver keep the heat down as low as possible without risk of "freezing" the buttons.* With the proper heat, the cupel is red, and the lead inside is distinct and glows strongly, the fumes rise plentifully, as shown in Fig. 2 (*a*), and toward the end of the operation,



(*a*) (*b*) (*c*)
FIG. 2

feathers or crystals of litharge gather around the sides of the cupel. If the heat is too high, the cupel appears white hot and glowing and the lead button is scarcely visible, while the fumes are very thin and rise rapidly to the top of the muffle, Fig. 2 (*b*), and the lead may even boil. On the other hand, if the heat is too low, the fumes will be dense and heavy and sink to the bottom of the muffle, Fig. 2 (*c*); the litharge will form on the surface of the lead, too thick to be absorbed properly and not hot enough to volatilize, and with a tendency to crystallize or freeze over the surface of the lead.

27. Freezing Point of Lead.—The freezing point of lead-silver will give some idea of the proper temperature for cupellation. The eutectic alloy of lead and silver contains

4 per cent. of silver and melts at 303° C. The melting point of pure lead is 326° C. and of pure silver 926° C. Most lead buttons contain less than 1 per cent. of silver and would therefore melt between 327° C. and 303° C., and remain molten until the button had decreased in its original weight to the extent of the total lead in it. At this point, the temperature required to keep the bead molten and prevent its freezing will rapidly increase, and as the melting point of silver is 926° C. it must be slightly higher. It has been found that 700° C. is the best temperature for the rapid formation of litharge and its ready absorption by the cupel, from which it is evident that the heat must be raised toward the end of the cupellation to prevent the bead from freezing. For a gold button it should be higher than the melting point of gold (1,063° C.).

28. Frozen Buttons.—In case the buttons do freeze, they may be reopened by putting a little charcoal or a small piece of wood in the front part of the muffle and closing the door. The gases from the wood or coal will reduce the film over the button, and as soon as the buttons are opened the door of the muffle is to be opened and the wood or coal withdrawn. A small piece of wood or charcoal placed in the cupel will produce the same result, but is likely to cause a slight loss by spitting. The results from a button that has been frozen and then reopened are never thoroughly trustworthy, however, as the reopening usually volatilizes considerable silver. Care should therefore be taken to prevent freezing, and if the buttons show the least sign of it, the door should be closed and the heat raised until they are out of danger.

29. Sprouting.—Beads of nearly pure silver, or those containing less than 33 per cent. of gold, *sprout* at the moment of solidification. Just before the silver bead becomes solid it absorbs oxygen from the air, the maximum absorption being about twenty-two volumes. This oxygen is suddenly expelled when the bead solidifies, bursts through the thin shell of solidified metal, forming a cauliflower-like

growth on the bead. The sudden bursting of the bead frequently throws fine particles of silver some distance from the beads. Large beads are more likely to sprout than small beads. To prevent sprouting, cover the cupel while in the muffle with a hot, empty cupel and withdraw gradually from the hot part of the muffle to the front, and after it has cooled down to the temperature of the front of the muffle, take it out, still with the covering cupel on, and allow it to cool for some time before removing the cover. The results from a sprouted button should never be accepted, as there is almost certain to have been some loss in sprouting.

30. Losses in Cupellation.—When cupeling for silver alone or for silver-gold, it is necessary to watch the end of the cupellation and remove the cupel as soon as the bead has become dull. A heavy loss of silver commences if the bead is kept beyond that time in the muffle. The gold bead can be left some time in the muffle after the bead becomes dull without any appreciable loss.

It is essential to good cupellation that there should be a cool *drive* and a hot *blick*.

The bead when cold should be white or yellow according to the quantity of silver it contains. It should be round, but if flat it indicates the presence of foreign metals. The surface where the bead is attached to the bone ash should be crystalline, and the bead should cling firmly. If the bead is not firmly attached to the bone ash or if it has rootlets, the indications are that lead is in the alloy. The cupel after cupellation should be a light yellow when cold, and show a smooth, uncracked surface.

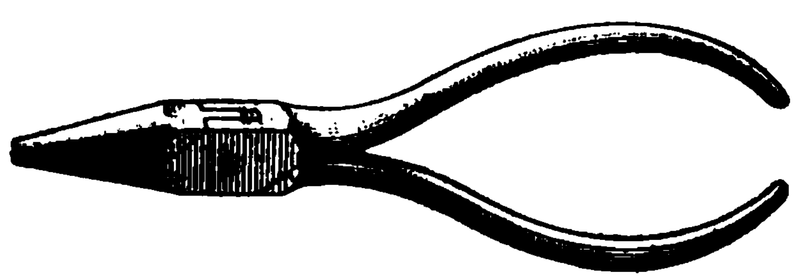


FIG. 3

31. Extracting the Bead.—As soon as the cupels are cool, the beads may be removed and weighed. For handling the small gold and silver beads, also called buttons, a small pair of pincers are necessary, the sharp-nosed style shown in Fig. 3 being preferable. The button is grasped firmly

with these and pulled loose from the cupel. Any litharge-soaked bone ash adhering to the bottom of the button is brushed off with a stiff tooth brush or with a small, cylindrical, double-ended button brush made especially for this purpose, but which is hardly as handy as the tooth brush.

32. Weighing the Buttons.—Before starting to weigh, the button balance must be carefully leveled up and adjusted. The scale pans are first removed (use the weight forceps for this purpose; the pans of the button balance must never be handled with the fingers), turned upside down, and shaken by tapping the back of the forceps, to shake out any dust or traces of gold left from previous weighings, and then replaced in their hangers. Then, having adjusted the balance, place the button to be weighed in one pan; sufficient weights are placed in the other pan to balance the button within the limit of the smallest weight of the set, and the balancing is finally completed with the rider (or if the button weighs less than the smallest weight of the set, the balancing is done entirely with the rider).

Having found the weight of one button, mark it down, and, leaving the weights in the pan, remove the button which has been weighed, clean the duplicate button, and place it in the pan which formerly contained the one already weighed. If the buttons are of equal weight, the second one will exactly balance the weights already in the pan; if not, the difference can be made up by shifting the rider or with the use of small weights, but, as a rule, a difference of as much as 1 mg. is altogether too much unless the ore is extremely high grade in silver or gold. For example, suppose an assayer wishes to weigh two duplicate buttons. The balance is in accurate adjustment, the beam on each side is divided into fiftieths, and he is using a 1-mg. rider. The first button is placed in the left-hand pan, and he finds that by placing his 10-mg., 5-mg., and two 2-mg. weights in the other pan, he slightly overbalances the button, while if he replaces one of the 2-mg. weights by his 1-mg. weight, the button side is heavier. Hence, the weight of the button

must lie between 18 and 19 mg. Leaving the 18 mg. in the pan and using the rider on the right-hand side, he finds that he must move it out 32 divisions to exactly balance the weight of the button; hence, he must add $\frac{32}{50}$ mg. or .64 mg. to the sum of the weights in the right-hand pan to obtain the weight of the button; consequently, button No. 1 weighs 18.64 mg. The same result would be obtained by leaving the 19-mg. weights in the right-hand pan and moving out the left-hand rider until the beam balanced, which it would do in this case with the rider at 18 divisions, and subtracting from 19 mg. the amount indicated by the rider, thus

$$19 - .36 = 18.64 \text{ mg.}$$

As a rule, it is best to have all the weights come as additions; that is, if the button is in the left-hand pan, it should not be quite balanced by means of the weights, so that the weight indicated by the rider will always be added, as this may avoid mistakes. Having obtained and recorded the weight of the first button, both by reading the weights in the pan and that indicated by the rider and the weight indicated by the empty spaces in the weight box and the rider, the operator next removes the button and cleans its duplicate and places it in the left-hand pan, and finds on testing the balances that No. 2 is somewhat heavier than No. 1. In other words, it overbalances the weight, and hence to balance it he moves the right-hand rider out slightly. He finds that by moving it out 7 divisions from the point it formerly occupied, the weight is exactly balanced; hence, the weight of the second button is $18 + \frac{39}{50}$, or 18.78 mg. Both buttons are then dumped into their parting capsules, the rider is returned to its place and the weights to the weight box, and the assayer is ready to weigh the next set of buttons. Some operators prefer to balance the buttons against each other, and finding the difference by means of the rider, add or subtract it from the weight of the first button to obtain the weight of the second button; but both this system and the use of the rider on the opposite side of the scale beam from that on which the weights are placed are more or less confusing where rapid work is required. As it is no uncommon

thing for the gold-silver assayer in a large works to be required to run from 30 to 40 samples a day, accurately determining both the gold and silver contents of each, it is evident that he must so systematize his work that everything will be read as directly and systematically as possible. In other words, if the weights are all additions there is never any danger of making mistakes, while if some of the weights are obtained by addition and some by subtraction, there is always a possibility of error.

33. Parting.—The buttons from the cupellation contain both the gold and the silver in the ore. To determine the exact amount of each, the buttons must be treated with dilute nitric acid, which dissolves the silver and leaves the gold behind as a black spongy mass or powder, which assumes its characteristic yellow color on heating. This operation is called "parting." The gold thus obtained is weighed up, and its weight, subtracted from the weight of the gold-silver buttons, gives the weight of silver in the ore. Duplicate buttons, unless very large and containing a considerable proportion of gold, may be parted together, in order to get a larger amount of gold to weigh and thus reduce the liability of error.

34. The operation of parting, in detail, is as follows: The buttons, if of any size, and particularly when containing much gold, are first flattened in a diamond mortar or by a blow of the hammer on an anvil, in order to expose as much surface as possible to the action of the acid. They are then replaced in a capsule, which is next filled about half full with dilute nitric acid of about 1.2 sp. gr. (made by mixing equal parts of strong chemically pure nitric acid and distilled water), and placed on an iron plate or piece of asbestos, or in a platinum triangle, over a Bunsen burner or alcohol flame. If there is enough silver in the button, it will at once commence to dissolve; if not, the acid will not attack it, and it will require to be first "inquarted." Assuming that there is a sufficient proportion of silver, the button will rapidly dissolve, the heat not being allowed to reach the

boiling point until all action has ceased and the solution has become colorless. If there is no gold in the buttons, they will not blacken on adding the acid, and will dissolve completely, leaving no residue. The parting may be stopped at this point and the ore reported as containing no gold.

If, however, the button blackens on adding the acid, and a black, spongy residue, or even a small black speck is left on the bottom of the capsule or floating around in the solution after action has ceased, the solution is brought to a boil for a minute; the capsule is then removed from the heat, and the gold, if it has broken up, should be collected by gently tapping the side of the capsule and giving it a rotary motion. Very small particles of gold may become enveloped in a film of air, and thus buoyed will float around on the surface of the acid solution, and refuse to be sunk by tapping the crucible. In such a case, "churn" or stir the contents of the capsule vigorously with a glass rod; this will collect the floating particles, and they may then be sunk by tapping. All black particles in the capsule must be assumed to be gold until proved otherwise, and if *any* black specks show, the parting should be continued through to the end, even if there is good reason to suspect that the specks are dirt. A little extra work is many times preferable to false assay returns.

35. Having gotten all the gold together at the bottom of the capsule, pour off the acid very carefully (preferably into another porcelain crucible, so that any gold which might escape through accident may be seen and recovered), using a glass rod to guide the stream and keep it from running back down the outside of the capsule. Then add fresh acid,* and boil for about 3 minutes. This removes practically the last traces of silver. The gold is collected as before, the acid poured off, and the gold washed three times with hot water. The washing is done by filling the capsules

*The practice in the Western mining states is to have the second acid of the same strength as the first. Many assayers prefer to use two strengths of acid for parting—a solution of about 1.16 sp. gr. (10 parts of strong acid to 16 of water) for the first heating, and another of 1.26 sp. gr. (16 parts acid to 10 of water) for the second.

with water, allowing the gold to settle, and then pouring the water off very carefully. Remove the last drops of water with a strip of blotting paper, being very careful not to take up any of the gold. Then heat for a moment over a Bunsen burner, to drive off the last of the water, and finally place in the muffle or over the blast lamp and bring to a good red heat. The gold will have assumed its natural yellow color, organic matter will have been burned off, and by the aid of a magnifying glass any impurities can be readily distinguished. This heating is called "annealing."

The annealed gold sticks together and can be transferred to the scale pans of the button balance for weighing. The assayer must take great care to get all the gold out of the capsule, and the weighing must be done with extreme care.

36. Instead of doing both parting and annealing in the porcelain capsules, many assayers prefer to part the buttons in test tubes, and then, after washing the gold, to transfer it to the capsules or to small porous "annealing cups." This makes it impossible, however, to part more than one set at a time, as the test tube must be held over the flame. The gold is transferred from the test tube to the capsule or annealing cup as follows: The test tube containing the gold is filled to the brim with water; then, the capsule is placed, upside down, over the mouth of the test tube, and with a quick motion the whole is inverted, so that the test tube is standing bottom upwards in the capsule. The test tube is drawn up until its mouth is nearly level with the rim of the capsule. The water is held up in the test tube by the pressure of the outside air on the surface of the small amount of water that has flowed out into the capsule, and the gold settles rapidly to the bottom of the capsule. As soon as the last black specks have settled, the test tube is raised very gently until its mouth is level with the brim of the capsule, and then with a quick jerk slid off sidewise, the water falling out as soon as the tube is clear of the capsule. The water is then drained off from the gold in the capsule, as previously described, and the gold is annealed and weighed.

37. Distilled water and nitric acid used for parting silver-gold beads must be tested to see that it is free from chlorine, or *HCl*. To test for the presence of chlorine, add a drop of silver-nitrate solution to the suspected water or acid, and if a cloudiness, caused by the precipitate of silver chloride, results, chlorine is present. To free nitric acid from chlorine, add silver-nitrate solution, a drop at a time, shaking the acid after each addition and allowing the precipitate of silver chloride to settle. When a cloudiness no longer results from the addition of silver-nitrate solution, the nitric acid should be filtered or decanted off, to remove the precipitate of silver chloride, and the pure acid transferred to a reagent bottle ready for use. Chlorine may be removed from distilled water in the same manner, but usually water containing chlorine is thrown away, it not being considered worth the time required to eliminate the chlorine by $AgNO_3$, for a fresh lot can easily be distilled.

38. Inquartation.—If a gold-silver bead does not contain at least $2\frac{1}{2}$ times as much silver as gold, the parting acid will not completely dissolve out the silver, or it may not attack the bead at all, and it becomes necessary to add sufficient silver to bring the proportion in the bead up to this figure. This addition of silver to beads is called "inquartation," from the fact that it was formerly believed that gold and silver could not be parted by acid if a larger proportion of gold than one-quarter was present.

Chemically pure (c. p.) silver foil is used for inquartation. A small bead is usually alloyed by merely melting it up on charcoal, with a small piece of silver foil, by the blowpipe. Larger beads are wrapped up, together with the necessary amount of silver foil, in sheet lead, and the whole cupeled down in the usual manner. The inquarted beads are flattened and then parted as usual.

If the amount of silver in the bead is not much over three times that of the gold, the bead will not break up in parting, but the silver will dissolve out and leave a black skeleton or sponge of gold of the same shape as the original

bead. On annealing, this sponge will shrink somewhat and turn yellow, but will hold together and retain its shape, making it very convenient for weighing. For this reason, as small an excess of silver as possible is added for inquartation. The experienced assayer can usually tell by the color of a bead very nearly what proportion of gold it contains, and he will then weigh out enough silver to bring this proportion down to about 1 to 3.

39. The weight of the gold subtracted from the original weight of the beads leaves the weight of the silver in the beads. Thus, suppose the beads used in this case are parted and found to contain together 2.46 mg. of gold; subtracting this from the combined weight of the two beads (37.42 mg.) the result is 34.96 mg., the weight of silver in the two beads. The average weight of the silver in the beads is, therefore, $\frac{34.96}{2}$, or 17.48 mg., and the average weight of the gold is $\frac{2.46}{2}$, or 1.23 mg. Then, if ore charges of $\frac{1}{10}$ A. T. were used, each milligram weight in the bead

TABLE II

COLOR TEMPERATURES

Color of Bead	Degrees Centigrade	Degrees Fahrenheit
Lowest red visible in the dark	470°	878°
Dark blood red or black red	532°	991.6°
Dark red, blood red, low red	566°	1,050.8°
Dark cherry red	635°	1,175°
Cherry red, full red	746°	1,374.5°
Light cherry, light red	843°	1,549.4°
Orange	900°	1,652.0°
Light orange	941°	1,725.8°
Yellow	1,000°	1,832.0°
Light yellow	1,080°	1,976.0°
White	1,205°	2,201.0°

represents 10 ounces per ton; and the ore assays 174.8 ounces of silver and 12.3 ounces of gold per ton.

40. Scale of Color Temperatures.—There are a number of scales of color temperature: that given in Table II is from White & Taylor's paper in the Transactions of the American Society of Mechanical Engineers.

FIRE-ASSAY FOR LEAD

41. Fire-Assay for Lead.—In most respects, the fire-assay for lead is very similar to the crucible assay for gold and silver. In making an assay for lead it would be illogical to add litharge to the fluxes, as that would increase the size of the button and give an uncertain result. The fluxes with the exception of litharge are practically identical with the general fluxes used for gold and silver assays, and if litharge is added can be used for such assays.

42. Lead Fluxes.—The fluxes for lead ores are made up in accordance with the kind of ores to be assayed. With a little experience, the assayer will be able to judge what mixture to make in order to slag his ore; in general, however, either of the stock fluxes given will successfully flux most lead ores, without any changes or additions.

		Parts by Weight
No. 1.	Sodium bicarbonate, $HNaCO_3$	4
	Potassium carbonate, K_2CO_3	4
	Borax glass, $Na_2B_4O_7$	2
	Flour	1

		Parts by Weight
No. 2.	Sodium bicarbonate, $HNaCO_3$	13
	Potassium bicarbonate, K_2CO_3	10
	Borax glass, $Na_2B_4O_7$	5
	Flour	2½ to 4

If the ore contains sulphur, the proportion of flour should be reduced, and with lead sulphide concentrates flour may be omitted.

From 1 to 4 tenpenny nails should be added to the charge for a sulphide before the salt or borax cover.

43. Lead-Assay Charges.—The general practice in the United States is to use 5 g. of ore and run duplicates. The method is as follows: Two 5-g. charges of ore are mixed in 10-g. clay crucibles with from 15 to 20 g. of lead flux apiece (for 10 g. of ore use 30 g. flux); the assay is then covered with borax and the fusion is made in the muffle furnace. Put the crucibles into the muffle when it is at a low red heat, and then gradually raise the heat to a full red at the finish. This will avoid danger of boiling over and will give higher results than if the assay is run very hot, as there will be less loss from volatilization. In about 20 to 30 minutes all "cooking" will cease and the charges subside to a quiet, liquid fusion. The heat is then raised or the crucibles are set back into the hotter part of the muffle, and they are left in for a few minutes longer, in order that the slag may become thin and fluid. They are then removed, tapped gently on the edge of the furnace to collect the lead, and poured into molds.

44. The Button Treatment.—As soon as the fusions in the molds are cool, they are removed, the slag is broken away from the buttons, and the buttons are hammered out flat, or if large may be hammered into cubes. The hammering will free them of slag. They should be soft and malleable. If they are brittle, they contain sulphur, arsenic, antimony, bismuth, or some similar element; copper or iron makes the buttons hard; *any* impurity makes the buttons heavier than they should be. The slag should be clean and brittle, and should contain no shots of lead.

The buttons are brushed and weighed. They should agree within about ½ per cent. (25 mg. on 5-g. charges). The weighing is done on the pulp balance, or, better, on the analytical balance, as no such great delicacy is required as to necessitate the use of the button balance. The assay is reported in per cent.—that is, the ore contains so many per cent. of lead. The figuring is very simple, particularly if

5-g. charges are used. The weight of the buttons is added and the sum divided by the total weight of ore taken, giving the percentage of lead in the ore. For example, if the buttons from two 5-g. charges of ore weigh 3.73 g. and 3.75 g., respectively, the lead contents of the ore are:

$$\frac{3.73 + 3.75}{5 + 5} = \frac{7.48}{10} = .748$$

or the ore contains 74.8 per cent. of lead.

In a ton of 2,000 pounds, 74.8 per cent. of lead is equivalent to 1,496 pounds, as 1 per cent. is 20 pounds. If lead is worth $4\frac{1}{2}$ cents a pound, then the value of the lead in the ore is $67.32.

If the ore runs very high in silver—several hundred ounces per ton—the buttons should be cupeled and the amount of silver determined and deducted from the weight of the lead. Small amounts of silver may be neglected.

45. Volatilization of Lead.—To assay lead by the fire-assay and get good results requires very careful work and considerable practice. There is invariably more or less lead lost through volatilization, and the results of the assay are, consequently, always somewhat lower than the actual contents of the ore, unless the button contains impurities. How to keep the loss as small as possible is the problem confronting the assayer. If the blue flame above the crucibles "jumps," the muffle is too hot, and, unless the temperature is immediately lowered, the assay will be defective. After the blue flame ceases, the crucible is allowed to remain in the muffle a short time and the heat raised just before pouring. If the heat is too high, considerable lead is volatilized; if it is too low, on the other hand, the assay must be kept in the furnace longer, from 1 to 2 hours, and as slow volatilization is constantly going on, the ultimate result is apt to be the same as though the heat were too high. The proper heat and time must be determined by experiment. A number of assays of the same ore with the same charge should be run under different conditions, at various temperatures, and for different lengths of time at the same temperature. The

highest result is in all probability the most nearly correct, provided the button is pure and malleable; and the conditions under which it was obtained should be adopted for general work.

46. Lead Sulphides.—The charge given in Art. 44 is for oxidized ores—carbonates, oxides, sulphates, etc. If the lead is in the form of a sulphide (galena) or is associated with other sulphides, iron nails or wire should be added to the charge, as in the crucible assay of sulphide ores for gold and silver, to take up the sulphur. Two tenpenny nails are usually sufficient. Potassium cyanide may be used as a desulphurizer instead of iron, but it is very dangerous to handle and is liable to reduce other metals than lead, so that most assayers prefer to use nails.

Lead assays may be run in the wind furnace if desired, although the muffle furnace is much more convenient. If run in the wind furnace, the crucibles are placed in the furnace while the fire is low, in order to get a gradual heat, and fresh fuel is piled around them. The fusion takes from 15 to 25 minutes. As soon as they are quiet they may be removed and poured, and the buttons cooled, beaten out, and weighed as before. The heat in the wind furnace is not so readily controlled as in the muffle furnace, and the results of lead assays run in this way are consequently less uniform and reliable than those run in a muffle furnace.

EXAMPLES FOR PRACTICE

1. Ore charges, 5 g. each. Weights of buttons, 2.47 and 2.5 g., respectively. What is the per cent. of lead in the ore?

Ans. 49.7 per cent.

2. Ore charges, 10 g. each. Weights of buttons, 6.93 and 6.89 g., respectively. What is the per cent. of lead in the ore?

Ans. 69.1 per cent.

FIRE-ASSAY FOR COPPER

47. Assay of Copper Ores for Gold and Silver. The assay for copper in ores is much more quickly and accurately performed by the wet method of assaying than by the dry method. When it is desired to ascertain the quantity of gold and silver in a copper ore or matte, it is usually better to employ the fire-assay. The preparation of the ore for assay is similar to that described for gold-silver ores; namely, crushing, bucking, sifting, and weighing. Several methods of making these assays have been suggested by different experimenters, most of them based on the quantity of copper in the ore.

48. Crucible Assay of Copper Ores.—The gold in a copper ore can be determined by crucible assay if the copper does not exceed 15 per cent. of the ore. Assume that the copper ore taken for assay weighed $\frac{1}{2}$ A. T., or, say, 15 g., and that the ore contained 10 per cent. of copper, in oxide form. Then, $15 \times .10 = 1.5$ g. of copper, but metallic copper is but 63 per cent. of copper oxide; therefore, $\frac{1.5 \times 100}{63} = 2.38$ per cent. of copper oxide in the ore. From Balling's table, the parts of silica required to form a copper monosilicate are .379 and $.379 \times 2.38 = .9$ g. silica required to flux the copper oxide. The remainder of the charge can be taken from a general formula, for example:

Litharge	20 g.
Sodium bicarbonate	20 g.
Flour	1.6 g.
Borax cover.	

49. Copper-Sulphide Ore.—In case there is less than 15 per cent. of copper in a sulphide ore, nails may be used for desulphurizing or a small quantity of niter. The button obtained is scorified with silica, test lead, and borax to remove the copper, after which the button from the scorifier is cupeled, weighed, parted, annealed, and weighed. The operation differs little from the regular crucible assay for gold.

50. Fire-Assay of Low-Grade Copper Ores.—If the ore is low grade, a number of assays are made, the resulting buttons scorified to remove the copper and reduce them to the proper size for cupeling. In case the ore contains sulphur, it is roasted to remove the sulphur, and after it is cool ammonium carbonate is added. The charge in this case should be:

Ore	$\frac{1}{2}$ A. T.
Litharge	1 A. T.
Sodium bicarbonate	1 A. T.
Silica	$1\frac{1}{2}$ A. T.
Salt cover.	

In case the ore is not roasted, the sulphur must be oxidized by niter, using about 25 g. for $\frac{1}{2}$-A.-T. ore and a salt cover.

51. Scorification Assay of Copper Ores or Mattes. In this case there are two separate assays: one for gold and one for silver.

In the case of gold, ten scorification assays are made in 3-inch scorifiers with the following charge:

Ore	.1 A. T.
Test lead	70 g.
Silica	1 g. to save the scorifier
Borax glass	1 g. for slag
Test lead cover.	

After fusion, the slag is poured from all scorifiers, and the buttons rescorified, with an addition of test lead and a little borax. Two scorifiers are combined and poured into molds and the buttons from .2 A. T. of ore are placed in one scorifier with 50 g. of test lead and some borax. These buttons are scorified to about a 30-g. button, then cupeled, and parted in the regular way.

52. Assay of Matte for Silver.—The assay for silver is a combination method. Four portions of ore or matte weighing each .5 A. T. are placed in 18-ounce beakers, and are dissolved in 90 c. c. of water and 25 c. c. of nitric acid. When the action becomes slack, add an additional 25 c. c. of concentrated nitric acid. When the copper is all

dissolved, add 5 drops of concentrated hydrochloric acid, stir vigorously with a glass rod, and allow the solution to stand over night to permit the silver chloride to settle. Filter through double filter, wash two or three times, burn filter on 2-inch scorifier, and add 25 g. of test lead, and a pinch of borax glass, scorify until half covered, pour the 15-g. button, cupel, and weigh.

FIRE-ASSAY OF TELLURIDES

53. Crucible-Assay of Tellurides.—Tellurium minerals, such as petzite, sylvanite, and calaverite, are readily decomposed by litharge, with the formation of a lead-silver-gold alloy and tellurous anhydride, TeO_2. The scorification assay for tellurides is now seldom used, owing to the loss that occurs by reason of volatilization. The crucible assay is far more reliable, and, when properly carried out, as accurate as the wet assay.

The best results from experiments were obtained with a crucible charge as follows:

Ore	½ A. T.
Litharge	3 A. T.
Sodium bicarbonate	½ A. T.
Borax glass	5 g.
Salt cover.	

Neither niter nor iron is used as an oxidizer and during the earlier stages of firing the temperature is to be kept low, or until the gases have nearly ceased escaping from the melt. The buttons obtained are rather large, owing to the desire to collect all the gold in the ore, and in all cases they should be soft and malleable.

54. Assaying Telluride Slags and Cupels.—With the charge given, the loss of gold in the slag was very little; however, tellurium ores new to the assayer should have an assay run of the slag, to ascertain if loss occurs. It has been found by experiments made that the greatest loss occurs during cupellation, and that this loss is both by absorption and by volatilization.

When cupellation takes place at a low temperature, with the formation of considerable feathered litharge, the loss by volatilization is so small that it may be disregarded; but when cupellation takes place at a higher temperature, volatilization is materially increased.

The charge used by Hilderbrand and Allen for assaying slag from telluride assays was:

Litharge	1 A. T.
Argol	2 g.
Slag	½ A. T.
Salt cover.	

The charge for assaying cupels from the cupellation of lead-tellurium buttons was:

Litharge	2 A. T.
Soda	1 A. T.
Borax glass	1.5 A. T.
Argol	2 g.
Cupel material	½ A. T.
Salt cover.	

CORRECTED ASSAYS

55. Assay of Slags.—In case rich ores are assayed, some gold and silver is likely to be left in the slag, for which reason the slag is pulverized and remelted with suitable fluxes. Ores that contain much ferric oxide or many metallic oxides are likely to retain gold and silver in the slag owing to their high fusibility. This may be explained by stating that lead, which is the collecting agent, sinks through the pulp and pasty slag, before the ore has become so fluidly slagged that the lead can reach the gold. Ores containing silver will, if they contain zinc, arsenic, antimony, and tellurium, leave some silver in the slag.

56. Assay Charges for Slags.—Charges should vary according to kind of slag assayed, and this can be fairly determined by the color of the slag. If one A. T. of ore was taken for assay, the pulverized slag if basic should have silica added and be thoroughly mixed with 1 A. T. of litharge,

1 g. of flour, and a borax cover. If the ore was acid, then silica is to be omitted and a small quantity of soda added to the charge given. Scorification slags need only a little flour and borax.

57. Assay Charges for Cupels.—To assay a cupel, remove and pulverize the saturated part. The phosphates present in the bone ash make the slag from cupels pasty, but this can be corrected by fluorspar or borax. The pulverized cupel is mixed with 50 g. of litharge, 30 g. of sodium bicarbonate, and 30 g. of borax glass. The mixture is fused in a crucible, poured, and the resulting button cupeled.

In some cases, the cupel and slag are pulverized and fused together, suitable fluxes being used for the purpose.

PLATINUM ASSAY

58. Fire-Assay of Platinum.—The following are given as the shortest accurate methods for the determination of platinum without the use of an oxyhydrogen blowpipe. When carefully conducted, they compare favorably with the long and tedious wet methods. The material to be assayed may be conveniently divided as follows:

1. Low-grade platinum ores, often containing considerable quantities of iridosmium.
2. Russian ores, with about 80 per cent. platinum. These are treated with mercury before they are sold.
3. Platinum alloys, containing silver, gold, etc.
4. Platinum foil, crucibles, etc. often containing iridium, etc.

In the first case, the ore must receive a preliminary treatment to concentrate the platinum; this can be done by panning down, where the platinum is in grains, or by fusion of 2 or 4 A. T. with litharge, etc., as in the crucible assay of a low-grade gold ore, which collects the platinum in a lead button.

The chief difficulty in the assay of Russian ores is to secure an accurate sample. To obtain this, melt a large quantity of the ore, 20 to 50 g., with six times its weight of test lead until the platinum has alloyed with the lead; this may be done in a scorifier; then pour, detach the slag carefully, as

the resulting button is brittle, and weigh the lead-platinum alloy. Pulverize in a mortar and weigh out portions for assay. When the assay is finished, the platinum in the lead must be recovered.

59. First Method for Platinum, Gold, Iridium, and Iridosmium.—Take the lead button or platinum alloy of such a quantity as shall contain about 100 mg. or platinum, and scorify at a high heat with 40 g. of test lead until the slag covers over; then pour into a mold, and when cold hammer to detach the slag. The button should weigh about 10 g., and will be malleable if it contains less than 5 per cent. of platinum.

Place the button in a large beaker with 200 c. c. of nitric acid 1.08 sp. gr. and heat until all action ceases; then filter through a small ashless filter paper, and wash the residue of lead, gold, platinum, iridium, etc. with water once or twice, transfer the paper to a porcelain capsule, dry, and then ignite in the muffle with the door open for 10 minutes to oxidize the lead remaining with the gold, etc. After cooling, heat to boiling with 1.08 sp. gr. nitric acid for several minutes, decant, wash thoroughly in the capsule, dry, anneal, and weigh the gold, platinum, iridium, and iridosmium.

Replace in the capsule, and warm with dilute aqua regia (1:5) for a few minutes, which readily dissolves the gold and platinum in the finely divided state in which they are left. Decant into a small beaker, ignite, and weigh the residue.

To separate iridium from iridosmium, boil the residue with strong aqua regia, decant, and wash; the final residue is iridosmium.

Evaporate the filtrate containing gold and platinum just to dryness to remove chlorine and nitric acid, but do not bake, as chloride of gold will be reduced; dissolve in water, add a few drops of hydrochloric acid, and precipitate the gold by warming with crystals of oxalic acid for half an hour, filter, and weigh the gold—or, better, cupel with silver (three times the weight of gold) to remove filter ash—part with nitric acid as usual, and weigh gold after annealing. This

treatment would also remove any trace of platinum. The platinum is estimated by difference, or may be precipitated as $(NH_4)_2PtCl_6$ after destroying the excess of oxalic acid.

If an ore contains *Pt*, iridosmium, *Os*, *Ir*, *Pd*, *Ru*, *Rh*, *Ag*, *Au*, *Cu*, *Fe*, and SiO_2, the scorification with lead removes iron and silica, as well as most of the copper, and volatilizes some osmium. The first treatment with 1.08 nitric acid dissolves most of the lead, all of the silver, copper if present, palladium, and some rhodium, leaving *Pb*, *Pt*, *Ir* (*Ru*, *Rh*, *Os*?), *Au*, and iridosmium. The second treatment with nitric acid after ignition takes out the remaining lead. Dilute aqua regia dissolves gold and platinum, leaving iridosmium, iridium, and some of the *Ru*, *Rh*, and *Os* present.

60. Second Method for the Determination of Base Metals and Silver, Gold, Platinum, and Iridosmium. This method takes advantage of the solubility of platinum in nitric acid when alloyed with at least 12 parts of silver.

The button or platinum alloy, containing from 100 to 200 mg. of platinum, is cupeled with sufficient silver to prevent freezing—this should be five or six times the platinum present—the heat should be very high at the end, and the button should be allowed to remain in the muffle for several minutes after brightening to remove lead as completely as possible, even if some silver is lost. After allowing for the silver added, the loss in weight is base metal. The button is hammered as usual, rolled out, and the silver dissolved by boiling for several minutes with concentrated sulphuric acid; the residue is washed, annealed, and weighed; the loss minus the silver added gives the silver; the residue is gold, platinum, iridosmium, etc. To the residue twelve times its weight of silver is added and the mixture cupeled with lead; the resulting button is hammered, rolled out, and parted with nitric acid 1.16, then with 1.26 sp. gr.

One such treatment will not remove all the platinum, so that the operation must be repeated or the boiling with acid prolonged until the residue no longer diminishes in weight. Unless this precaution be observed the gold will be too high.

The residue of gold, iridosmium, etc. is weighed and the loss from the previous residue determined as platinum. It is then treated with dilute aqua regia to dissolve the gold, and the final residue of iridosmium is obtained.

In this method, no account is taken of the other metals always associated with platinum. Palladium dissolves with the silver; iridium is left with the iridosmium if the aqua regia used is dilute (1 to 5) and only heated gently; if stronger acid is used, the iridium will dissolve with the gold.

61. Remarks on Platinum Assays.—Some very pure platinum foil assayed by the first method gave 99.70 per cent. of platinum.

The following results were obtained on an alloy containing lead, silver, gold, platinum, iridium, and iridosmium:

Wet Method: Pt, as $(NH_4)_2PtCl_6$, 10.53 and 10.30 per cent.; Au, by oxalic acid, 3.42 and 3.424 per cent.

First Method: Pt, 10.44 and 10.43 per cent.; Au, 3.456 and 3.45 per cent.; Ir, .5 per cent.; $IrOs$, .55 per cent.

Second Method: (Residue not retreated):

	Platinum Per Cent.	Gold Per Cent.
Ratio of silver to residue, 13 to 1 .	10.02	4.88
Ratio of silver to residue, 19 to 1 .	10.20	4.70
Iridosmium	.22	

These results are inserted to show the necessity for the modifications described under the second method.

Platinum if present in an ore to the extent of a small percentage of the gold and silver will not be noticed unless specially tested for, as it dissolves in the nitric acid with the silver. If present in greater quantity the button will freeze in cupellation before all the lead has been oxidized, and the bead will have a white, frosted appearance, sometimes resembling a cauliflower.

The addition of cadmium to an alloy containing platinum and silver renders the platinum insoluble in nitric acid. This fact is made use of for the determination of silver in gold bullion containing platinum.

CONTROL ASSAYS

62. The price paid by mills, smelters, or sampling works for ores is fixed or controlled by the result of assays run independently, by different assayers, on duplicate samples from each lot of ore. Such assays are aptly named *control assays*, or, briefly, *controls*. The only way in which they differ essentially from ordinary assays is in the extreme care taken in sampling and assaying, in order to insure perfect justice to both the shipper and the works. The following are the details of the method commonly adopted in the Western mining states.

63. Sampling Controls.—When the ore comes into the mill or smelter, the entire lot is crushed, if necessary, and a sample of from 5 to 10 tons, according to the size of the lot, is cut out, usually by an automatic sampler, or by quartering, or by the use of the split shovel. (Small lots of ore are not cut down, but the entire lot is treated in the same way as the sample from a larger lot.) This sample is further cut down in the same manner to 200 or 300 pounds.

This last sample is crushed quite fine by fine-crushing rolls or by a sample grinder, and is then cut down to about 2 or 3 pounds by quartering or by the use of a tin sampler, or riffle. This is the final sample. The whole of this sample is ground down to 100 mesh and screened.

The pulp is very thoroughly mixed by rolling on a rubber mixing cloth, and the sample is then divided into four equal parts by quartering or by the riffle. These are the control samples, and should weigh from 8 to 12 ounces each. They are put into separate sample envelopes, of which each mill has its own printed form. Each envelope is sealed with the seal of the works, and is marked with the mine number, the name of the mine, and the mill or lot number. The shipper is then given his pick of the samples, generally taking two, another is kept by the mill for assay, and the remaining sample is stamped or marked with the word "Umpire" and retained by the mill, to be assayed by a third party in case

the assays of the shipper and the mill fail to agree within reasonable limits. The shipper sometimes also writes or stamps his name on the envelope containing the umpire sample. Some works divide their final sample into five parts, so that both the shipper and the works can have two samples, in addition to the umpire sample.

64. Assaying for Settlement.—The shipper takes one of his samples to the mine assayer, if the mine employs one, or, if not, to any reliable custom assay office, and has it assayed, while the mill assayer assays the sample kept by the mill. They then compare results, and if they check reasonably close—say within 3 or 4 points (a "point" is $\frac{1}{100}$ ounce, or 20 cents per ton, in gold) on an ore carrying $20 or more per ton in gold—they "split," or average, the results, and the ore is paid for on this basis. For example, if the mine assay shows the ore to contain 2.06 ounces of gold and the mill assay gives 2.02 ounces of gold per ton, the shipper is paid for the average, or 2.04 ounces of gold per ton.

If, however, the two assays do not check within these limits, both assayers repeat their work. If, after repeating, they still disagree, and particularly if the mine assay is high, the umpire sample is sent to some disinterested and reliable assayer agreed on by both parties concerned, and his result is usually taken as final. If he does not check with either of the other assayers, however, the shipper may demand that the lot be resampled or may send his ore elsewhere, as it is always held until the assays are satisfactorily completed and the shipper has been paid for it, before treating.

65. Control Assaying.—The assays may be run by either the scorifier or crucible assay, using $\frac{1}{10}$, $\frac{1}{2}$, or 1-A.-T. charges, according to the grade and character of the ore. The assaying is done in the usual manner, but with great care. Many assayers prefer to use 20-g. crucibles for $\frac{1}{2}$-A.-T. charges, or to take double the number of $\frac{1}{4}$-A.-T. charges in 10-g. crucibles, in order to have room for a considerable excess of flux. Each assayer has his own way of checking

his work. A good scheme is to run three charges and part two buttons together and the third separately as a check. If the work has been well done, the gold from the two buttons should weigh almost exactly twice as much as that from the single button. If it does not, there is something wrong and the assay should be repeated. In parting, it is best to use the first acid quite weak—about 1 in 5—and the last acid strong.

The results of control assays are calculated as in ordinary assays, with gold at $20 an ounce and silver at the market price.

BULLION ASSAYING

BASE BULLION

66. Nomenclature of Bullion.—Bullion is uncoined gold and silver, or foreign coins that are to be remelted. In metallurgy considerable latitude is given to the word bullion; for instance, gold or silver amalgam is termed bullion; retort gold is termed bullion; pig lead from furnaces is termed **base bullion**; copper bottoms from copper furnaces that contain gold and silver is **copper bullion.** The silver-gold alloy resulting from the cupellation of argentiferous lead is termed **doré bullion.** Bullion in which gold predominates is termed **fine bullion,** and is the product resulting from refining doré bullion. The purity of bullion is denoted by parts in 1,000, that is, instead of saying a bullion is 99 per cent. pure, it is said to be 990 fine, meaning that out of 1,000 parts, 990 are pure metal and 10 parts impurities.

67. Base-Bullion Sampling.—In smelting gold and silver ores, over 8 per cent. of the charge is lead ore, and this collects the gold and silver, the same as litharge collects them in a crucible assay. Frequently, the lead and silver are very much in excess of the gold, and the lead very much in excess of the silver. In order to ascertain the value of the base bullion, it is necessary to sample and assay it for the

respective metals. The samples are taken from the bars of bullion by means of a steel punch similar to a harness-maker's punch, but larger and heavier. This punch is driven about half way through the bars, taking out cores about ¼ inch in diameter. Two sets of samples are taken from each bar, one from the top and one from the bottom, toward opposite sides and ends. When sampling a lot of bullion, the bars are usually sampled in bunches of five, as follows:

The five bars are laid side by side on the sampling platform, as shown in Fig. 4, and one sample is taken from each bar, starting at the outside upper or lower corner and working diagonally across, the samples being taken out at the points indicated by the solid circles. Then the bars are turned over, and a second set of samples taken out in the same order, but starting from the opposite end of the first bar and working along the other diagonal. In the figure, the positions of the punch holes in the lower sides of the bars are indicated by dotted circles.

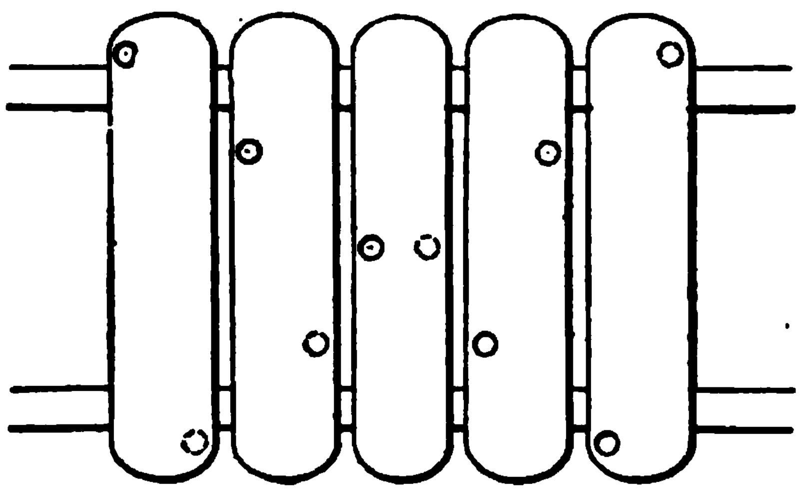


FIG. 4

68. The samples from the entire lot are melted up together in a clay or graphite crucible, care being taken that the heat does not rise to a point where the lead begins to cupel or scorify (volatilize), as any reduction in the proportion of lead in the bullion is equivalent to an increased proportion of precious metals, and the results would therefore be too high. As soon as the sample is perfectly fluid, it is thoroughly stirred with a clean iron rod, and then poured into a mold and cast into a thin, flat bar, 10 to 18 inches long, about 3 inches wide, and ½ or ¾ inch thick. The samples for assay are taken from this bar either by punching out pieces from the ends

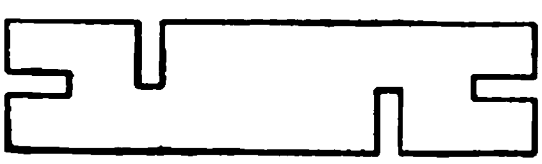


FIG. 5

and sides, as shown in Fig. 5, or by punching out pieces at intervals, diagonally across the bar, or cutting strips right across the bar. These samples should each be approximately of the weight required for the assay, usually ½ A. T. Four samples are usually taken from each bar.

69. Cupeling Base Bullion.—One-half A. T. is accurately weighed out from each sample. Each ½-A.-T. sample is then cupeled separately, the cupellation being conducted exactly as in ordinary ore assays, with as low a heat as is practicable. The cupels should always show "feathers." If the bullion is impure—if it contains considerable quantities of impurities besides lead, such as copper, arsenic, and antimony, or sulphur—it should be scorified down with a little borax, and if very impure, a little test lead, before cupeling. The cupels should be moved back into the hotter part of the muffle just before "blinking," and should then be withdrawn gradually, to prevent sprouting. A sprouted button should always be rejected.

As soon as the cupels are cool, the buttons are removed, brushed, and weighed. The weights should agree very closely—say within half an ounce on bullion running 200 ounces of silver to the ton.

70. Cornet Rolls.—The object of cornet rolls is to flatten the button so that it may be more readily attacked by acid when parting. They are not an absolute necessity but are a great convenience, for otherwise the button must be flattened out on an anvil. Braun's cornet rolls are 2¼ inches in diameter, with a face 1¼ inches wide, made of hardened steel, with the face ground perfectly true. They are operated by gears, and are adjusted to different thicknesses of cornets by the jackscrew above the rolls. The entire machine is 14 inches high, weighs 58 pounds, and is listed at $35. The feed-plates to and from the rolls prevent the cornet from dropping to the floor or bench.

71. Parting Bullion.—The parting may be performed in a porcelain capsule, but on account of the size of the button it is better to use a parting flask, of any one of the forms

shown in Fig. 6. Two buttons are placed in each flask and from 20 to 30 c. c. of dilute nitric acid, 1.16 sp. gr. (21° Baumé) added. This is boiled until the silver is dissolved, and then continued until the red fumes disappear. Shake the flask gently to collect the gold and then pour the acid off carefully, leaving the gold in the flask. Fresh acid of 1.26 sp. gr. (32° Baumé), or about 40 per cent. concentrated nitric acid, is placed in the flask and the acid boiled as before. By boiling a second time, the silver is more completely removed. It has been ascertained by Hilderbrand and Allen that more than two boilings in acid are necessary to eliminate all traces of lead and silver from a button. Miller states that by boil-

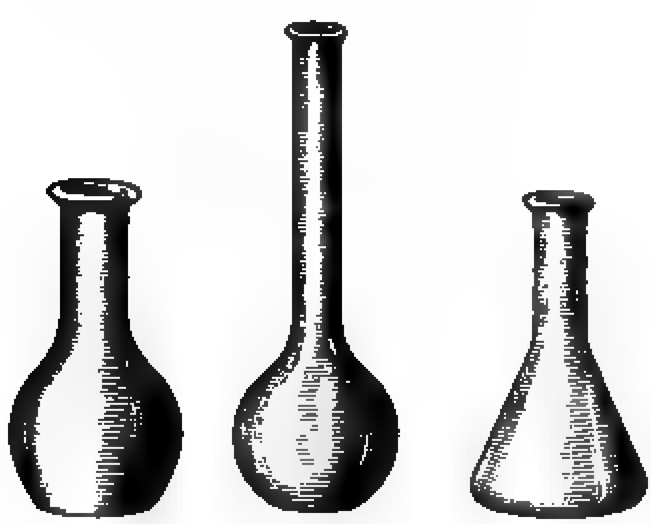


FIG. 6

ing in acid concentrated to 1.42 sp. gr. gold is dissolved in appreciable quantities. This would probably occur if the acid used was impure, that is, contained hydrochloric acid. However, it has not been definitely established that loss of gold occurs, although it has been definitely established that more than two washings and boilings are needed to obtain pure gold in bullion. If acid of 1.26 sp. gr. will not remove all the impurities from the bead, then stronger acid, either 1.35 or 1.46 sp. gr., should be used for the third washing. It was found by experiment that no decrease in the weight of the button occurred after the third washing, indicating that the bead was about pure gold. To prevent the acid from bumping and spurting during boiling insert a small glass

tube in the acid or a bit of charcoal or one or two charred beans. If the acid is heated nearly to boiling before the bead is dropped into it, the gold is not so apt to be broken up into fine particles, by the action of the acid.

After boiling, the gold is washed three times with distilled water, and is finally transferred to a porcelain capsule or an annealing cup, dried, annealed, and weighed. The weights of gold from the two sets of buttons should check very closely, and if they do not, two more samples should be assayed. Their results will usually check one or the other of the first two pairs. If they do not, the assay should be again repeated, and so on until good checks are obtained.

Base-bullion assays are always reported, like ordinary gold and silver assays, in *ounces per ton*. Fine-bullion assays are reported in parts per 1,000, or as so many "thousandths fine."

FINE-BULLION ASSAYING

72. Fine Bullion.—Fine bullion is composed mostly of gold and silver. If it contains more silver than gold it is termed doré bullion, or silver bullion. If, however, it contains more gold than silver it is termed gold bullion. Fine bullion is produced from retorting amalgam, from parting doré bullion, and from precipitates obtained in chlorination and cyanide mills, or other reduction processes. The gold in fine bullion is always determined by fire-assay. The silver may be determined either by fire-assay or by wet assay. The wet assay is adopted in the mints and in most large metallurgical works, as it is slightly more accurate and less troublesome than the fire-assay. It involves considerable knowledge of chemistry and very delicate manipulation. The fire-assay, if carefully done, will give very close approximate results and is the method commonly used by custom assay offices and small works.

73. Selecting the Sample.—The bullion is assayed from samples taken from bars, but a dip sample is taken in mints and refineries, sometimes for the assay and sometimes

for the check assay. The advantage of a dip assay over a bar assay is obscure.

The sample is obtained from the bar by chipping off the diagonally opposite upper and lower corners with a cold chisel or by boring to the center of the bar from the top and bottom, near the diagonally opposite corners, with a drill press or ratchet drill. The latter practice is the better, except in the case of bullion known to be very fine and uniform, and will detect any attempt at fraud, such as filling the center of the bar with lead or copper—a trick that is frequently tried. The very first borings should be rejected, as they are apt to be dirty and give low result. Borings are ready for immediate weighing. Chips have to be flattened out to $\frac{1}{32}$ inch on the anvil or in a small set of rolls made especially for the purpose, until they are thin enough to be cut up by the shears. The samples, borings, or rolled chips are placed in envelopes properly marked with the number stamped on the bar from which they are taken.

74. Weights for Bullion Assay.—The weights used for the gold assay of bullion are the $\frac{1}{2}$ g., which is stamped "1,000" and the decimal subsidary weights stamped 900, 800, 700, etc., and 90, 80, 70, etc. down to .5. These numbers denote the number of $\frac{1}{2}$ milligrams (*millièmes*) contained in the weight. Ordinary weights in the gram system may be used and each milligram will correspond to 2 millièmes in the assay system. Since the assay is reported in $\frac{1}{10000}$ part, the balance must indicate a difference in weight of .1 per 1,000, or .05, mg.

75. Preliminary Bullion Assay.—The approximate composition of the bullion is first determined by a preliminary assay. Half a gram of bullion is weighed out accurately on the button balance; this is wrapped in from 5 to 10 g. of pure lead foil and cupeled in a small cupel (weighing about 10 or 12 g.). The cupellation is conducted as in the base-bullion assay, with the same precautions, the cupels showing "feathers" just before finishing. The button should then be weighed and parted as usual.

The results indicated by the preliminary assay are used in making up the proof or correction assay, which should be as nearly as possible identical, in every particular, with regular assay charges. The amount of pure silver put into the proof charge is from 5 to 10 mg. greater than the amount indicated by the results of the preliminary assay, as it is roughly estimated that this amount is lost in the cupellation. If the bullion contains much copper, the amount should be determined by wet assay and a corresponding amount added to the proof. Small amounts of copper, however, may be disregarded. If the preliminary assay shows the bullion to contain too much gold to part without inquarting, sufficient pure silver is added to bring the proportions up to $2\frac{1}{2}$ parts of silver to 1 of gold. If the gold is not up to this proportion, some assayers add enough pure gold to make it so. This is not the general practice, however. The only advantage it gives is that the gold stays together in a cornet, and is consequently easier to handle, with less danger of loss during washing.

76. Platinum Parting Trays.—For parting the gold and silver in bullion assays, platinum parting trays are commonly used. The trays are simply small crates of platinum wire, divided into a number of compartments, in each of which is placed a small platinum cup—merely a small crucible—with a series of slits in the bottom to allow free circulation of the acid and permit it to drain off readily on removing the tray from the bowl. The gold-silver "cornets," made by rolling the buttons out into thin strips, annealing them by heating to redness in the muffle, and then twisting them into a spiral coil, are put in the tray, one in each cup, and the tray is then hung in a platinum bowl about 3 inches in diameter and 2 inches deep, filled to within about $\frac{1}{2}$ inch of the top with dilute nitric acid of 1.28 sp. gr. (50-per-cent. concentrated acid), and heated nearly to boiling. They are boiled 10 minutes in this acid; then this is poured off and the dish is refilled with fresh acid of the same strength (or sometimes the second acid is used

stronger), and they are boiled 10 minutes longer. The crate is then lifted out and the cornets washed in the crate with pure distilled water. They are then dried out over a Bunsen burner or an alcohol lamp, and the crate and its contents are then put into the muffle and allowed to come to a red heat to anneal the cornets. After cooling, the cornets are weighed.

77. Making the Proof.—Table III is used in making up the proof. The method of making up the proof and the use of the table are best illustrated by an example. Suppose a preliminary assay of 500 mg. of bullion gave 350 mg. of

TABLE III

Weight in mg. of Silver from Preliminary Assay of 500 mg. Bullion	Weight of Silver in mg. to be Used in Proof	Weight in Grams of C. P. Lead Foil to be Used in Proof and in Regular Assay	Per Cent. of Copper in Bullion as Determined by Analysis	Weight in mg. of C. P. Copper Foil to be Used in Proof
475	480	5	2.5	12.5
450	455–460	7	5.0	25.0
425	430–435	8	10.0	50.0
400	405–410	10	15.0	75.0
375	380–385	11	20.0	100.0
350	355–360	12	25.0	125.0
325	330–335	13	30.0	150.0
300	305–310	15	35.0	175.0
250	255–260	17	40.0	200.0
200	205–210	19	45.0	225.0
150	155–160	20	50.0	250.0

silver and an analysis showed 20 per cent. of copper. The table shows that from 355 to 360 mg. of pure silver are to be added, and also that 12 g. of c. p. lead foil will be required for cupellation. Now, as the bullion contains 20 per cent. copper, 100 mg. of c. p. copper foil and 50 mg. of c. p. test lead are also added. [The weight of test lead necessary in the proof is obtained by subtracting from the weight of bullion used the sum of the weights of silver and copper contained in the 500 mg. of bullion; viz., $500 - (350 + 100) = 50$ mg. of test lead.] The whole is wrapped in the 12 g. of c. p. lead foil, when it is ready for cupellation with the

regular assay. The proof is made up in this manner merely because the loss of silver in bullion during cupellation depends on the amounts of copper and lead present. The regular assay is performed as follows: Weigh out two portions of bullion of 500 mg. ($\frac{1}{2}$ g.) each; wrap in the proper amount of lead foil, as shown in the table, together with what pure silver or gold may be necessary to secure the proper proportions for parting, and cupel. The cupel containing the proof assay is placed between the two regular assays and run along with them. The resulting buttons are weighed, rolled out into ribbons, annealed, rolled into cornets, and parted in parting flasks or in a platinum crucible. The gold-silver buttons should check within 1 mg., and the gold almost exactly. The loss of silver in the proof assay should not be more than 5 mg., if the cupellation has been run properly; the gold loss, unless the bullion runs quite high in gold, will be hardly noticeable. The amount of the loss of the proof assay is carefully determined and a corresponding amount added to the results of the bullion assay, to make up for the loss during cupellation. The buttons should be bright and clean.

78. Calculating the Bullion Assay.—The result of the assay is reported in parts fine per thousand—that is, the bullion is reported to contain so many parts pure silver and so many parts gold, and the remainder is base metal, usually lead and copper, the whole summing up to 1,000 parts. For example, if the gold-silver buttons from two 500-mg. ($\frac{1}{2}$-g.) charges weigh 417 and 418 mg., respectively, and the proof assay shows a loss of 4.5 mg., the first button contains $\frac{417 + 4.5}{500} = \frac{843}{1{,}000}$, or 843 parts of fine silver and gold, and $1{,}000 - 843 = 157$ parts of base metals; and the second button contains $\frac{418 + 4.5}{500} = \frac{845}{1{,}000}$, or 845 parts fine silver and gold and 155 parts base metals; or, the average total fineness (silver and gold) of the bullion is 844. If the weight of the gold from the two buttons is 24 mg., this

weight divided by the weight of the bullion from which the gold was derived, 1,000 mg. (two $\frac{1}{2}$-g. charges), gives the contents of the bullion in gold. The bullion therefore contains $\frac{24}{1000}$, or 24 parts fine, of gold. This amount deducted from the total fineness of the bullion (844) gives the fineness in silver, which is thus found to be 820. The silver contents may also be found in the same way as the gold, by dividing the weight of silver in the buttons by the weight of bullion taken; thus:

421.5 + 422.5 = 844 mg., corrected weight of gold-silver buttons.

844 mg. − 24 mg. = 820 mg., weight of silver in 1,000 mg. (two $\frac{1}{2}$-g. charges) of bullion.

$\frac{820}{1000}$ = .820, or 820 parts silver in bullion.

The results of the preceding assay would be reported:

Silver	820 fine
Gold	24 fine

This would be stamped on the bar with steel dies as a decimal; that is,

Silver	.820
Gold	.024

Using two $\frac{1}{2}$-g. charges for assaying greatly facilitates the calculations, as the weights of the two charges sum up to 1,000 mg., and by adding the results from the two buttons together, each milligram is $\frac{1}{1000}$ or, 1 part, so that no division is necessary, the fineness in silver or gold being the same as the weight of silver or gold in milligrams.

DETERMINATION OF COPPER IN SILVER BULLION

79. If fine bullion contains much copper and is to be assayed by the fire-assay, it is necessary to know the proportion of copper present in order to make up the proof assay properly.

To determine the amount of copper, dissolve $\frac{1}{2}$ g. of bullion in dilute nitric acid (the bullion should be beaten or

rolled out so that it will dissolve rapidly) and add HCl in very slight excess. Test for an excess of HCl from time to time by adding a drop of the solution to a drop of silver nitrate solution. As soon as a white cloudiness results, the HCl is in excess. Then filter off the precipitate of silver chloride, make the filtrate alkaline with an excess of ammonia, and titrate for the copper as usual.

ASSAYING

(PART 4)

WET ASSAYING

SOLVENTS AND PRECIPITANTS

1. Volumetric Analysis.—Volumetric analysis consists in dissolving a substance in liquid chemicals, and then precipitating the substance by other liquid chemicals. The liquids mostly used for dissolving substances are acids, and those employed as precipitants are solutions of alkalies, or bases. In some cases, the reaction that takes place between the solutions is direct and may be readily calculated; in other cases, it is indirect and must be indirectly calculated. For example, if a solution is known to be of such a strength that 1 c. c. = .01 g. of *Fe*, the quantity of iron in a solution can be found by the number of cubic centimeters of the solution required to precipitate it; these figures are read off from a burette.

In the indirect reactions, the element sought is not precipitated as an element that can be calculated directly, but as a compound of known composition; for example, sulphur is usually precipitated as barium sulphate, and the quantity of sulphur is calculated from the formula $BaSO_4$. Again, in estimating the quantity of copper by the iodide method, the iodine liberated, when cuprous iodide is formed, is first estimated, and from this the copper is determined.

2. The Unit Volume for Calculation.—The unit volume adopted for volumetric calculations is the cubic centimeter (abbreviated to c. c.), which is equal to 1 g. of distilled water; hence, 1,000 g. of distilled water is equal to 1 liter. The strength of a solution depends on the number of grams of the chemicals contained in a liter, or 1 c. c. When the unit volume of the reagent required to saturate a unit volume of some other solution is known to have a certain strength, the element sought can be calculated. The strength of a reagent is determined by tests, rather more frequently than by calculation, owing to the difficulty encountered in obtaining absolutely pure chemicals. When a reagent is added gradually to a solution, certain physical and chemical changes become apparent as soon as there is the slightest excess of the reagent over the actual quantity necessary for combination. For example, in the permanganate method of iron analysis, a single drop of the permanganate solution, after all the iron is oxidized, will impart a characteristic pink color to the solution under examination; or, in analyses in which the reagent used is not in itself a powerful colorant, some chemical may be used as an *indicator*, with which the least excess of the reagent solution will give a strong and characteristic color. The point at which the coloration or change first appears is called the **end point.** The reagent solution is known as a **standard solution**; and the weight of the element in question, with which each cubic centimeter of the standard solution will combine, is called the **standard** of the solution. By multiplying the number of cubic centimeters of the standard solution used up to the end point by the standard of the solution, the amount of the element in question contained in the solution is obtained. This method is known as **volumetric analysis, or titration.**

3. Ores to be Assayed in the Wet Way.—Assayers are frequently called on to make determinations of other elements than gold, silver, and lead—elements that cannot be accurately determined by the fire-assay, and those that

require some knowledge of chemical analysis for their determination. For example, the price paid for ores by smelters depends on the amount of iron oxide, lime, and silica they contain. Manganese oxide acts, up to a certain point, like iron oxide, and the same premium is paid for both. Copper in ores is also paid for if present in any considerable quantity; hence, the assayer will find it greatly to his advantage to be able to perform the analyses for these substances. The wet determination of lead is also quite common in smelters and lead works. Zinc is an important, though undesirable, constituent of many silver and gold ores, and its presence and amount are likely to affect the value of the ore considerably. Copper is almost invariably analyzed in the wet way.

4. **Gravimetric Versus Volumetric Analysis.**—Volumetric analyses are usually considerably quicker and less troublesome than gravimetric analyses and fire-assays, and are therefore used wherever speed is essential. It must not, however, be inferred from this that volumetric determinations are not accurate. Any of the schemes given in this Section are accurate to a small fraction of 1 per cent., and the volumetric assay for some elements is more accurate than the gravimetric. There is rather more room for error in volumetric work, as a slight variation in the end point of the titration or a change in the strength of the standard solution may make a slight difference in the results; but such errors, if the chemist is careful and standardizes his solutions frequently, are so small that they may safely be neglected. If duplicate determinations are run, any considerable error could not very well escape notice. The average result of the duplicate determination is always taken. Duplicates are seldom run in gravimetric work, on account of the extra work involved and the small probability of error. The final result in gravimetric analyses is obtained by actually *weighing* the precipitate containing the element sought, and consequently, if the analysis has been properly conducted and no error has been made in the weighing and calculations

the result is practically absolute. A good chemist can make volumetric determinations check very closely with gravimetric work.

5. Preparation of Ores for Wet Assay.—In all analyses, ores, in order that they may be in the best condition to be acted on by acids, must be pulverized as finely as possible. This is accomplished by grinding or rubbing in an agate mortar, treating only a few grams of ore at a time. The pulverization should be carried on until the ore is in the form of an impalpable powder; that is, until no gritty feeling is noticeable when a small portion of the powder is rubbed between the thumb and fingers.

APPARATUS FOR WET ASSAYS

6. Beakers.—Glass beakers, which are necessary for holding solutions, are made of a thin, tough glass that will stand considerable heat, so that solutions may be boiled in them. Beakers come in "nests" of six, as shown in Fig. 1. The smallest, or No. 1, beaker will hold about 100 c. c., while the largest, or No. 6, has a capacity of 1 liter (1,000 c. c.). The form with the lip for pouring, as shown in the figure, is

FIG. 1

FIG. 2

most convenient for general work, but such beakers are somewhat more expensive than the form without a lip.

7. Casseroles.—Porcelain casseroles, Fig. 2, are used for dissolving ores in acids. They are particularly useful when the solution must be evaporated down to dryness, as with care they will stand this operation without any danger

of cracking, whereas beakers would be very liable to crack as soon as the solutions in them are boiled dry. Casseroles come in various sizes. The 2-, 3-, or 4-ounce sizes are most convenient for ordinary work, as the ore can be well covered without using an excess of acid.

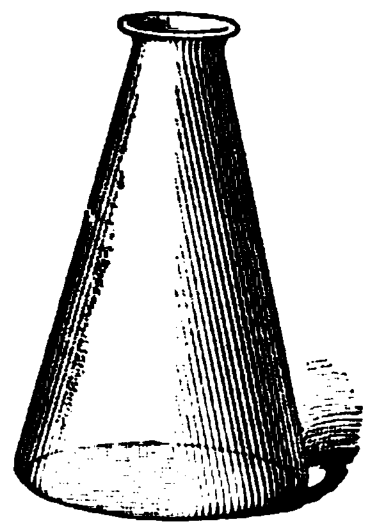


FIG. 3

8. Flasks.—Flasks of the shape of the wash bottle shown in Fig. 10, made of the same kind of glass as the beakers, are useful for various purposes, such as for dissolving ores, receiving solutions, etc. The 4-, 8-, and 16-ounce sizes are most convenient.

Fig. 3 illustrates an Erlenmeyer flask, which is very handy for precipitating from solutions and for general analytical work, on account of the large flat bottom that is exposed to the action of the hot plate or sand bath, and owing to the fact that any precipitate forming in the solution in the flask has a tendency to fall away from the sides and on to the bottom of the flask. Small flasks of this shape are used in the copper determination.

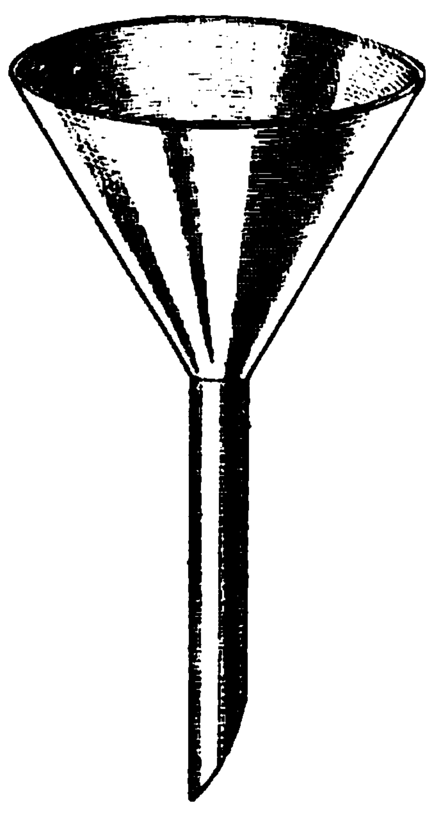


FIG. 4

9. Funnels.—Glass funnels are necessary for making filtrations. The angle between the sides should be 60°, and the stem should be ground off at an angle, as shown in Fig. 4, to draw the stream off to one side, thus lessening the capillary attraction between the tube and the solution, and consequently hastening the filtration.

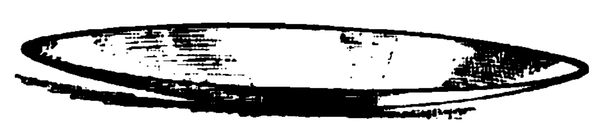


FIG. 5

10. Watch Glasses.—Watch glasses, Fig. 5, are very useful for covering beakers, casseroles, and funnels, and for receiving weighed charges of ore or chemicals, precipitates, etc. Like the beakers, watch glasses come in assorted sizes.

11. Burettes.—Burettes are graduated glass tubes fitted with glass stopcocks, as shown at *a*, Fig. 6, or with rubber-hose connections and pinch cocks, as shown at *b*, from which standard solutions are run into solutions to be titrated or tested volumetrically for certain elements. Burettes come in 25-, 50-, and 100-c. c. sizes graduated to $\frac{1}{10}$ c. c. The 50-c. c. size is most convenient.

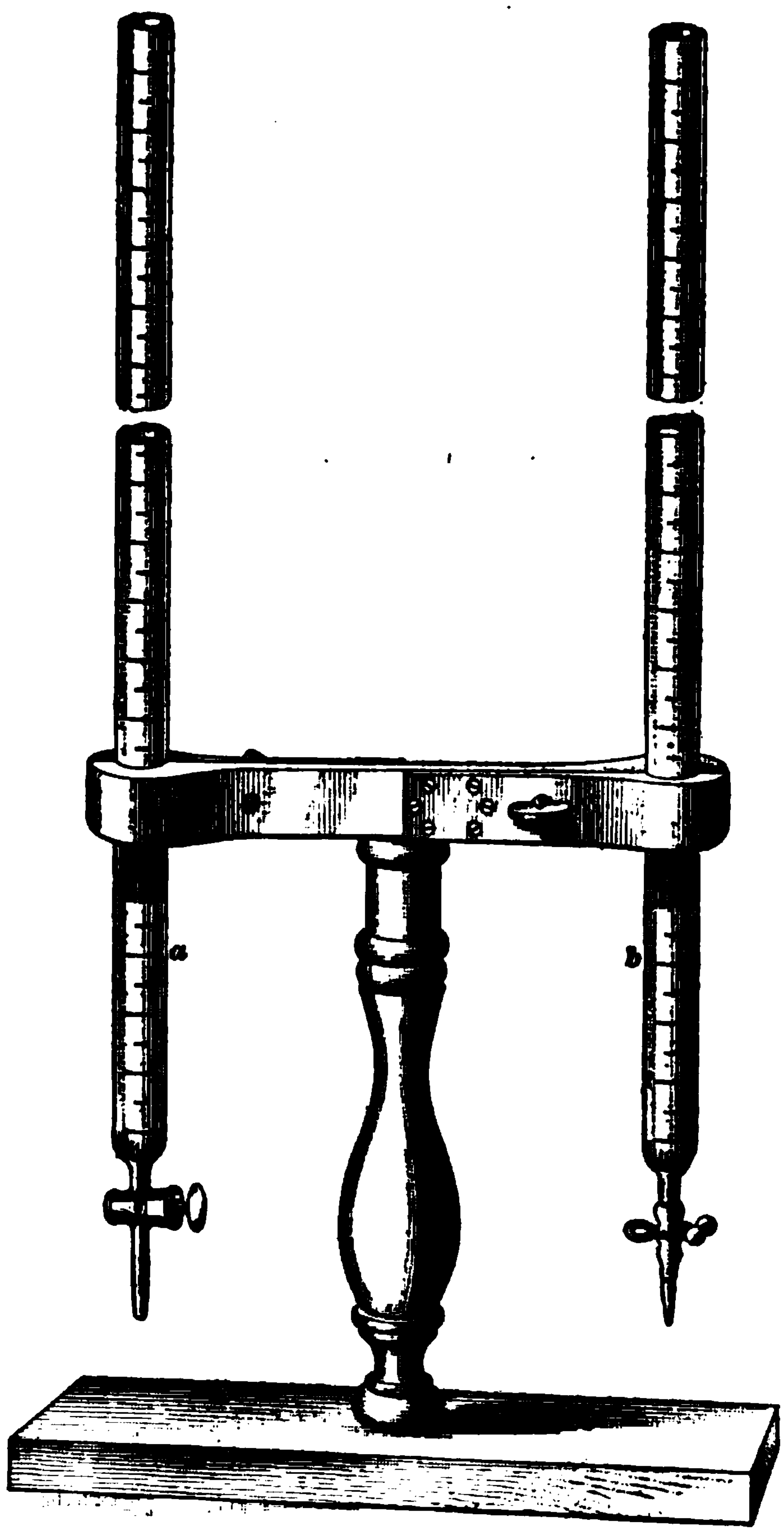


FIG. 6

12. Spot Plate. The spot plate, Fig. 7, is an oblong plate of white porcelain, 5 in. × 6 in., with a number of small depressions in its surface for receiving the indicator solutions used in some titrations. A few drops of the indicator solution are put into each depression by the use of a dropping tube; then, as the standard solution is run into the solution under examination, the latter is tested from time to time for excess of the standard solution, by taking a drop out on the end of a stirring rod and adding it to the indicator solution in one of the depressions. As soon as the standard solution is in the

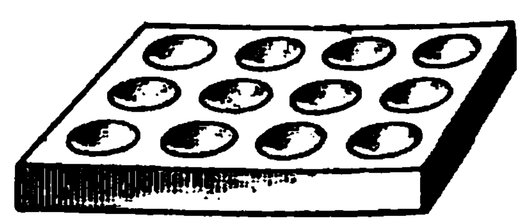


FIG. 7

slightest excess over the amount necessary to convert the element sought in the solution under examination, a drop of the latter solution will cause a characteristic reaction when added to the indicator solution.

13. Crucibles.—Porcelain and platinum crucibles are necessary for making fusions and igniting (heating at a high heat) precipitates.

14. Graduates.—Glass graduates are necessary for measuring acids, etc. They may be had in various shapes and sizes. A graduate with straight sides, as shown in Fig. 8, is most convenient. Glass graduates may be had in sizes varying from 50 to 500 c. c.

15. Titrating Dish.—A flat, shallow, white porcelain dish of about 1 quart capacity—an ordinary ironstone porcelain vegetable-dish answers very well—is very convenient for making titrations, as the end point shows sharply against the white porcelain. A sheet of white paper behind a beaker will serve the same purpose.

FIG. 8

16. Filter and Burette Stands.—Wooden or iron stands are necessary for holding funnels and burettes while filtering and titrating. A wooden burette stand is shown in Fig. 6. The filter stand is somewhat similar, but has conical

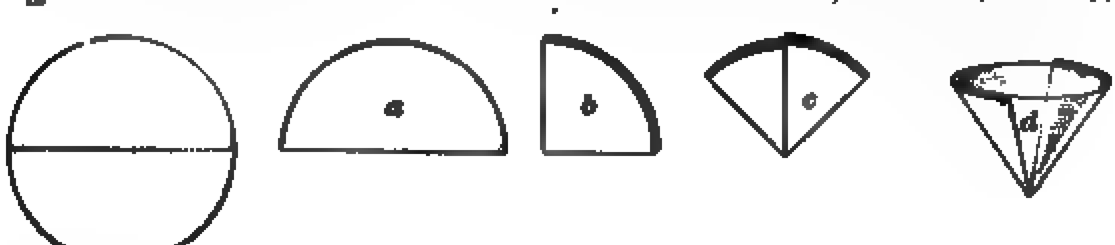


FIG. 9

holes cut in the cross-bar for the funnels. Sometimes, racks are used that will hold several funnels.

17. Filter Paper.—Filter paper is tough, porous paper, used for filtering solutions. It may be obtained in circular sheets of various sizes, in packages of 100. The sheets are folded to fit into the funnel. The folding is done as follows:

Fold over along the diameter, as at *a*, Fig. 9; fold again, corner to corner, as at *b*; then open out into form *a* again, fold one corner and outside edge in to the center line, and turn the paper over and fold the other corner in the same way on the other side, as at *c*. The filter will then open out into the form *d*, which fits exactly into the funnel. After placing it in the funnel and moistening with water to make it stick to the sides of the funnel, it is ready for filtering.

18. For use in quantitative work, where the filter paper has to be burned or ignited with the precipitate, it is necessary that the weight of the ash of the filter should be known. There are two methods of accomplishing this. One is to employ what are known as *ashless*, or *acid-washed*, *filters*. These are filter papers that have been washed with hydrochloric and hydrofluoric acids, thus removing the solid portion of the ash and leaving practically nothing but carbon in the filter paper, so that it will burn without leaving any ash. The other method is to determine accurately the weight of the ash of the filter paper, and then to subtract this weight from the amount obtained after igniting each precipitate on its filter paper. Such filter papers can be bought in packages of 100, with the weight of the ash that each sheet will produce stamped on the back of the package. Where very accurate work is desired, the chemist can determine the weight of his own filter papers by burning three or four down to a white ash in an accurately weighed porcelain or platinum crucible. The crucible containing the ash is then weighed, and the increase in weight over the weight of the crucible alone is the weight of the ash. This amount divided by the number of filter papers used gives the weight of the ash from each filter. Ashless, or acid-washed, filters can be tested in the same way, and three or four of them should give such a small amount of ash that it would not make any perceptible increase in the weight of the crucible.

19. Wash Bottle.—The chemist has constant use for distilled water. The water for immediate use is kept in a large flask or wash bottle, of from 16 to 32 ounces capacity,

with two tubes passing through the cork and arranged so that on blowing in one the air pressure forces the water up and out through the other, which is drawn out to a fine jet at the tip. Fig. 10 shows the wash bottle and the arrangement of the tubes. The air tube need only extend through the cork. The long water tube should extend nearly to the bottom of the flask, so that it will remain under water even when the water in the flask gets very low. A good plan is to stop the glass tube an inch or two from the bottom, and then put on a short piece of rubber tubing, as shown in the figure, extending to the bottom, or barely clearing it. This avoids the risk of pushing the tube through the bottom of the flask when putting in the cork. A flexible joint of rubber tubing at the jet, as shown, is also convenient for directing the stream. Ordinary flasks may be used for wash bottles, but specially made flasks, with a heavy ring around the mouth to bear tight corking, are stronger and better. The neck may be wrapped with twine for handling when the water is hot. When boiling water in a wash bottle, the cork should always be loosened and set up on the edge of the mouth, otherwise the pressure of the steam will force the water out through the jet, little by little, the air tube being so small that the steam will not escape fast enough to keep the pressure in the flask down to atmospheric pressure.

Fig. 10

20. Stirring Rods.—Glass stirring rods, of assorted lengths and sizes—say from 3 to 8 inches long and from $\frac{1}{8}$ to $\frac{3}{16}$ inch in diameter—are essential in the laboratory. A short piece of rubber hose on the end of the rod will prevent it from being pushed through the bottom of the beaker. The rods may be bought of the proper size, or the chemist may buy the glass in 3- or 6-foot lengths and make his own rods. To break the glass rod, make a scratch with a file at the point where it is desired to break it, grasp the rod with both hands, one hand on each side of the file mark, close the

thumbs together, on the side of the rod opposite the mark, and break by pressing up with the thumbs. The broken ends may be made round and smooth by heating with the blow-pipe or in the blue flame of a Bunsen burner. Stirring rods may also be made from glass tubing by closing the ends of the tubing in the Bunsen flame or with the blowpipe flame. A glass rod with a piece of rubber on the end is called a **policeman**, and may be used for removing precipitates from beakers or other dishes.

21. Bunsen Burners, Tripods, Etc.—Bunsen burners and tripods with wire gauze and asbestos cloth tops are essential for heating water and solutions. Every laboratory, moreover, should be supplied with an exhaust hood, under which all boiling with acids should be done. The hood is simply a small chamber connected by a flue with the outside air, to draw off disagreeable and poisonous fumes and prevent their spreading through the laboratory. An iron heating table or "hot plate" and a large gas burner are necessary under the hood if much work is to be done, and are very convenient under any circumstances.

22. Sink and Slop Jar.—Every laboratory should have a sink and faucet in connection with or convenient to the working desk. A 5-gallon earthenware jar should be set under or alongside the desk, to receive washings, spent solutions, etc.

REAGENTS FOR WET ASSAYS

ACIDS AND SOLVENTS

23. Acetic Acid.—Acetic acid, $H_4C_2O_2$, is obtained from the distillation of wood, and for assay use should be 50 per cent. chemically pure. It is employed, in the titration of lead, and for acetate solutions needed in volumetric and gravimetric analyses.

24. Hydrochloric Acid.—Hydrochloric acid, HCl, should be obtained chemically pure, concentrated to 1.2 sp.

gr. Dilute acid is used mostly in the laboratory; it contains 1 part chemically pure concentrated acid and 2 parts distilled water. It is used for many purposes as a solvent and as a precipitant.

25. **Nitric Acid.**—Nitric acid, HNO_3, should be chemically pure, concentrated, and have a specific gravity of 1.42. Dilute acid consists of 1 part acid and 2 parts distilled water. Pure nitric acid gives up its oxygen readily. When it acts on metals it forms nitrates, metal atoms being substituted for the hydrogen of the acid. If the substance on which the acid acts has not the power to replace the hydrogen, the action consists in oxidation. In the reduction of nitric acid, nitrogen peroxide, NO_2, nitrous acid, HNO_2, nitric oxide, NO, nitrous oxide, N_2O, and ammonia are formed, according to the conditions under which the reaction takes place. The oxides N_2O and NO_2 are readily changed to NO, which is the reaction that commonly takes place when treating metals with nitric acid. Nitric acid is a powerful solvent and oxidizing, as well as a desulphurizing, agent.

26. **Nitro-Hydrochloric Acid** (*Aqua Regia*).—One volume of concentrated nitric acid added to 3 volumes of hydrochloric acid forms aqua regia, which should be prepared only as required. It may be used either concentrated or dilute. It is better for dissolving gold when slightly diluted.

27. **Oxalic Acid.**—Oxalic acid, $H_2C_2O_4$, is obtained in chemically pure crystals. When diluted with water it is a weak solvent. It is principally used in the form of ammonium oxalate. One gram of the crystals to 10 c. c. of water makes a practically saturated solution of oxalic acid.

28. **Sulphuric Acid.**—Sulphuric acid, H_2SO_4, chemically pure and concentrated, with a specific gravity of 1.82, finds considerable use in the laboratory. It is a powerful solvent and precipitant. The dilute acid is prepared by adding 1 part of acid to 5 parts of water. The dilute acid is used for precipitating barium: 1 c. c. will precipitate .4291 g. of barium as sulphate Concentrated sulphuric acid will not

attack iron, but the dilute acid will. Dilute sulphuric acid only slightly attacks lead. When diluting, the concentrated acid must always be poured into the water—never water into the concentrated acid. As the union of sulphuric acid and water produces heat, an explosion might result if water were poured into the acid.

29. Sulphurous Acid.—Sulphurous acid, H_2SO_3, can be obtained in a chemically pure solution. It is not as active a solvent as sulphuric acid, nor is it used nearly so often in the assay office.

SALTS

30. Ammonium Chloride.—Ammonium chloride, NH_4Cl, is used with ammonia as a precipitant of lime, iron, etc. It is best used as an aqueous solution having 1 g. of the salt dissolved in 8 c. c. of water.

31. Ammonium Molybdate.—Ammonium molybdate, $(NH_4)_6Mo_7O_{24}$, known as **molybdate solution**, is used as a precipitant for phosphorus and arsenic. In making up the solution, 1 g. of molybdenum trioxide, MoO_3, is dissolved in 4 c. c. of ammonia. This solution is poured into 10 c. c. of nitric acid having the specific gravity of 1.2. The resulting solution is warmed to 45°, and 1 c. c. of a 10-per-cent. solution of crystallized sodium phosphate is stirred in vigorously. This solution is allowed to stand over night before using.

32. Ammonium Nitrate.—Ammonium nitrate, NH_4NO_3, is a good oxidizing agent that is readily decomposed by heat.

33. Ammonium-Hydrogen Sulphide.—Ammonium-hydrogen sulphide, $(NH_4)HS$, is used as a precipitant for iron, manganese, nickel, cobalt, and zinc. It is prepared by passing hydrogen sulphide, H_2S, into dilute ammonia. It loses its strength rapidly in air, and must be kept in a stoppered bottle, and be prepared fresh from time to time. It is a powerful solvent of the oxides, and of the sulphides of arsenic, antimony, and tin. The precipitate is a sulphide of the metals named.

34. Ammonium Acetate.—Ammonium acetate, $NH_4C_2H_3O_2$, is made by adding strong acetic acid to strong ammonia until the solution is just acid, and then a few drops of ammonia sufficient to make the solution alkaline. It is a powerful solvent of lead salts.

35. Ammonium Oxalate.—Ammonium oxalate, $(NH_4)_2C_2O_4$, is used principally as a precipitant for calcium. In making up the solution, 1 g. of salt is added to 10 c. c. of water. One c. c. of this solution will precipitate .0145 g. of CaO.

36. Ammonium Sulphide.—Ammonium sulphide, $(NH_4)_2S$, is prepared by conducting hydrogen-sulphide gas, H_2S, into a bottle two-thirds full of concentrated ammonium hydrate, NH_4OH, until it is saturated, which is indicated by the bubbles coming from the liquid undiminished in size. The bottle is then filled with concentrated ammonia and the solution thoroughly mixed. This stock solution should be kept in full tightly stoppered bottles, and the bottles should be colored, as light decomposes the ammonia sulphide. Before using, the stock solution should be diluted with twice its volume of water, the diluted solution being kept in the ordinary colored-glass reagent bottle. Arsenic, antimony, and tin are soluble in this solution. The solution will precipitate aluminum as white aluminum hydrate, $Al(OH)_3$; chromium as greenish chromium hydrate, $Cr(OH)_3$; iron as black ferrous sulphide, FeS; nickel as black nickel sulphide, NiS; cobalt as black cobaltous sulphide, CoS; manganese as pink manganous sulphide, MnS; and zinc as white zinc sulphide, ZnS.

37. Chlorine or Chlorine Water.—Chlorine, Cl, may be generated by treating bleaching powder (chloride of lime, $CaOCl_2$) with sulphuric acid; the gas may be absorbed in water. Chlorine water must be kept in a colored-glass bottle or in the dark, for in the light the chlorine will decompose water and form hydrochloric acid, HCl. Chlorine gas may also be prepared by mixing 50 g. of coarse salt and 40 g. of powdered black oxide of manganese, and adding to it

when cold a mixture of 125 g. of concentrated sulphuric acid and 60 g. of water; they should be well shaken together and warmed, and the gas as it comes over gently collected in water contained in a black-glass bottle.

38. Yellow Ammonium Sulphide.—Yellow ammonium sulphide, $(NH_4)_2S_2$, is made by adding a small quantity of flowers of sulphur to ammonium sulphide and shaking until the sulphur is dissolved. Enough sulphur should be added to give the solution an amber color.

39. Barium Chloride.—Barium chloride, $BaCl_2$, is principally used as a precipitant for sulphur trioxide, SO_3. If 1 g. of the pure salt is added to 10 c. c. of distilled water, 1 c. c. of the solution will precipitate .0327 g. of SO_3.

40. Barium Carbonate.—Barium carbonate, $BaCO_3$, has the power to unite and form insoluble compounds with some metallic sesquioxides. It is used to separate iron from other metals, like manganese, that do not unite with it. Barium carbonate may be prepared by precipitating a pure barium-chloride solution with ammonium carbonate, the mixture then being washed on the filter until all the ammonia salts have been removed. The wet precipitate should be stirred into the water so as to form a thin cream, or emulsion. It should be thoroughly mixed before using.

41. Barium Hydrate.—Barium hydrate, $Ba(OH)_2$, may be prepared by dissolving barium oxide in the proportion of 1 g. of salt to 10 c. c. of water. This should be digested or heated for several hours, and then the pure liquid filtered off and kept in a well-stoppered bottle.

42. Magnesia Mixture.—Magnesia mixture is made by dissolving 1 part of magnesium chloride, 2 parts of ammonium chloride, and 4 parts of ammonia in 8 parts of water. The mixture is allowed to stand several days and is then filtered. It is used as a precipitant for phosphorus and arsenic.

43. Bromine Water.—Bromine water, $Br + H_2O$, may be formed by making a saturated solution of bromine in

distilled water. It should be kept in a tightly stoppered colored-glass bottle and in a cool place. When opening the bromine water bottle in warm weather, care should be taken, for there is liable to be a sudden rush of vapor on withdrawing the stopper, and this vapor is not only disagreeable, but somewhat poisonous.

44. Calcium Hydrate.—Calcium hydrate or lime water, $Ca(OH)_2$, may be prepared by slaking fresh quicklime and adding to it a large quantity of water. The mixture is placed in a large glass bottle, shaken several times, and then allowed to settle. The clear solution is decanted off and used as a reagent. It contains 1 part of lime and several hundred parts of water.

45. Sulphureted Hydrogen.—Hydric-sulphide, or sulphureted hydrogen, H_2S, is formed by treating iron sulphide, FeS, with sulphuric acid. If iron sulphide cannot be obtained, it may be prepared by fusing iron nails with sulphur, in the proportion of about 1 part by weight of iron to 2 parts by weight of sulphur. H_2S gas may be led into water until the water is saturated, the saturated water being then used as a reagent. The water should be kept in colored-glass bottles, as it is quickly decomposed when exposed to the light. When it is desired to precipitate any substance from the solution by means of H_2S, it will be better to conduct the gas itself into the solution than to employ water charged with the gas, on account of the fact that, in order to add a sufficient amount of gas, it would be necessary to add a very large amount of water, thus unnecessarily increasing the bulk of the solution.

Hydrogen sulphide will precipitate the following metals from their salt solutions: Cadmium, as a white cadmium sulphide, CdS; bismuth, as a black bismuth sulphide, Bi_2S_3; copper, as a brownish black cupric sulphide, CuS; lead, as a black lead-sulphide precipitate, PbS; mercury, as a reddish brown mercuric sulphide, HgS; silver, as a black silver sulphide, Ag_2S; arsenic, as a yellow arsenious sulphide, As_2S_3; antimony, as an orange antimonious sulphide, Sb_2S_3; and tin,

as a brown stannous sulphide, SnS, or as a yellow stannic sulphide, SnS_2.

46. Platinic Chloride.—Platinic chloride, $PtCl_4$, is used occasionally as a precipitant of potassium sodium and lithium. The solution is prepared by dissolving platinum in aqua regia, evaporating to dryness, and dissolving in 1 c. c. of hydrochloric acid and 9 parts of water. One c. c. of this solution will precipitate .048 g. of K_2O.

47. Potassium Bichromate.—Potassium bichromate, $K_2Cr_2O_7$, is a powerful oxidizing agent, and is used in volumetric analysis. One part of potassium bichromate will convert 6 parts of ferrous iron into the ferric state. It is much used in titration.

48. Potassium Chromate.—A potassium-chromate, K_2CrO_4, solution is made by dissolving 1 g. of salt in 10 c. c. of water. Potassium chromate will precipitate yellow barium chromate, $BaCrO_4$; yellow strontium chromate, $SrCrO_4$; brick-red silver chromate, Ag_2CrO_4; red mercurous chromate, Hg_2CrO_4, from mercurous-salt solutions, and yellow mercuric chromate, $HgCrO_4$, from mercuric-salt solutions; yellow lead chromate, $PbCrO_4$; yellow bismuth chromate, $BiCrO_4$; and brown basic cupric chromate.

49. Potassium Iodide.—Potassium iodide, KI, is made into a solution by dissolving 1 g. of salt in 25 c. c. of water.

50. Potassium Ferrocyanide.—Potassium ferrocyanide, $K_4Fe(CN)_6$, is known as *yellow prussiate of potash*. It is used in solution when titrating for iron. With a ferrous solution it gives a white precipitate, which quickly turns blue; with a ferric solution it gives a dark blue precipitate; with a nickel solution, a greenish white precipitate; with a cobalt solution, a green precipitate; with a manganese solution, a white precipitate; and with a zinc solution, a white precipitate of zinc ferrocyanide. Potassium ferrocyanide gives a white precipitate from silver-, mercury-, lead-, bismuth-, and cadmium-salt solutions. Copper ferrocyanide

is thrown down as a reddish brown precipitate from copper-salt solutions when potassium ferrocyanide is added to them.

51. Potassium Ferricyanide.—Potassium ferricyanide, $K_3Fe(CN)_6$, known as *red prussiate of potash*, is made into solution by adding 1 g. of the salt to 10 c. c. of water. With ferrous salts, potassium ferricyanide gives a dark blue precipitate, and with ferric salts a reddish brown precipitate. With a solution of nickel salt, yellowish brown nickelous ferricyanide, $Ni_3Fe_2(CN)_{12}$, is formed. Brownish red cobaltous ferricyanide, $Co_3Fe_2(CN)_{12}$, brown manganous ferricyanide, $Mn_3Fe_2(CN)_{12}$, and brownish yellow zinc ferricyanide, $Zn_3Fe_2(CN)_{12}$, are precipitated from their respective salt solutions. Potassium ferricyanide is used as an indicator on the spot plate, with the bichromate test for iron.

52. Potassium Permanganate.—Potassium permanganate, $KMnO_4$, has an intense purple or reddish purple color when in an aqueous solution. It is made into a solution in which 6.25 g. of the pure salt is dissolved in 1 liter of water, and is used in the titration of iron, manganese, phosphorus, and lime. It readily gives up its oxygen, converting *-ous* salts into *-ic* salts; the moment this reaction is complete there is a pink tinge imparted to iron solutions.

53. Stannous Chloride.—Stannous-chloride, $SnCl_2$, solutions are made by dissolving 1 g. of the salt in 3 c. c. of HCl and 8 c. c. of water. Metallic tin should be kept in solution, and should be kept from the air to prevent the formation of oxides. Stannous chloride is used in the volumetric determination of iron by the bichromate method, since it converts ferric to ferrous salts. As an indicator of gold in solutions, it furnishes a beautiful purple color, known as the *purple of Cassius*. From their salt solutions, stannous chloride precipitates: white silver chloride, $AgCl$; black mercurous chloride, and white mercuric chloride; white plumbic chloride, $PbCl_2$; and white cuprous chloride, Cu_2Cl_2.

54. Silver Nitrate.—Silver nitrate, $AgNO_3$, is a salt prepared by dissolving pure silver in nitric acid. If 1 g.

of the salt is dissolved in 20 c. c. of water, 1 c. c. of the solution will precipitate .0104 g. of chlorine. Silver nitrate is used for titrating potassium-cyanide solutions, the normal solution being 16.86 g. of $AgNO_3$ in 1,000 c. c. of H_2O; 1 c. c. of this solution is equal to .0129 g. of KCN.

55. Potassium Sulphocyanate.—Potassium sulphocyanate, $KCNS$, is made into a solution by dissolving 1 g. of the salt in 10 c. c. of water. When used to test ferric salts, it gives a blood-red coloration.

56. Sodium Salts.—*Sodium bromide*, $NaBr$, will precipitate silver from solutions of silver bromide.

Sodium chloride, $NaCl$, is used to precipitate silver from solutions containing silver.

Sodium acetate, $NaC_2H_3O_2$, is used as a salt, or in a 10-per-cent. solution, to precipitate iron and aluminum in the basic-acetate separation of these metals.

Sodium nitrate, $NaNO_3$, may be used for an oxidizing agent, in fusions, although, generally, the corresponding potassium salt is used in its place.

Sodium-ammonium-hydrogenphosphate, $NaNH_4HPO_4 \cdot 4H_2O$, known also as *salt of phosphorous* and *microcosmic salt*, is used in blowpipe and some gravimetric analyses.

Sodium thiosulphate, $Na_2S_2O_3 + 5H_2O$, known also as *hyposulphite*, or *hypo*, is used to dissolve silver from its ores, in the Kiss, Von Patera, and Russell lixiviation processes. It is also used as a standard solution in the determination of copper by the iodide method.

ALKALIES AND ALKALINE SALTS

57. Ammonia.—Ammonia, NH_4OH, is used in aqueous solution having a specific gravity of .96, and is obtained by diluting with 2 volumes of water the strongest concentrated ammonia having the specific gravity of .88.

Ammonium hydrate will precipitate from solutions the following hydrates: aluminum hydrate, $Al_2(OH)_6$, as a white flocculent substance; greenish blue chromium hydrate,

$Cr(OH)_3$; red-brown ferric hydrate, $Fe(OH)_3$; green nickelous hydrate, $Ni(OH)_2$; blue basic cobaltous salt; brown manganous hydrate, $Mn(OH)_2$; and white zinc hydrate, $Zn(OH)_2$. It will also precipitate brown silver oxide, Ag_2O; a black mercurous and a white mercuric ammonium compound; a white lead hydrate, $Pb(OH)_2$; a dark blue basic cupric fluid; a white bismuth hydrate, $BiO(OH)$; and a white cadmium hydrate, $Cd(OH)_2$.

58. Ammonium Carbonate.—The ordinary commercial ammonium carbonate, $(NH_4)_2CO_3$, known also as *sesquicarbonate*, produces in solution a mixture of the neutral and acid carbonates. This feature is objectionable when the neutral carbonate is to be used, and hence the solution is diluted by adding 1 c. c. of ammonium carbonate to 4 c. c. of water. This alkaline salt is of a readily decomposable nature, and will precipitate white magnesium carbonate, $MgCO_3$; white barium carbonate, $BaCO_3$; white strontium carbonate, $SrCO_3$; and white calcium carbonate, $CaCO_3$. The salt is a desulphurizing agent, and is used for decomposing sulphate of copper.

59. Potassium Hydrate.—To make a solution of potassium hydrate, KOH, 1 g. of the alkaline salt is dissolved in 10 c. c. of water. The salt is known as *potash lye* and *caustic potash*. It may be substituted in most cases for sodium hydrate, since it produces similar chemical compounds.

60. Potassium Carbonate. — Potassium carbonate, K_2CO_3, is an alkaline carbonate that finds considerable use in fusion assays, and in gravimetric analyses. The anhydrous salt is pulverized, and forms a strong alkali.

61. Sodium Hydrate.—Sodium hydrate, $NaOH$, also termed *sodium hydroxide*, is used as a 10-per-cent. solution; that is, 1 g. of the hydrate is dissolved in 10 c. c. of water. The alkali will precipitate barium, calcium, magnesium, and strontium as white hydrates having the formula $R(OH)_2$. It will also precipitate white aluminum hydrate, $Al(OH)_3$; white ferrous hydrate, $Fe(OH)_2$; white zinc hydrate, $Zn(OH)_2$.

greenish blue chromium hydrate, $Cr(OH)_3$; red-brown ferric hydrate, $Fe_2(OH)_6$; green nickelous hydrate, $Ni(OH)_2$; a blue basic cobaltous salt that turns violet in an excess of ammonia; and brown manganous hydrate, $Mn(OH)_2$. Sodium hydrate will precipitate silver, mercury, lead, copper, bismuth, and cadmium, either as oxides or as hydrates, and antimony and tin as hydrates.

62. Sodium Carbonate.—Sodium carbonate, Na_2CO_3, is used pulverized and dry for fusions, and in saturated solutions as a precipitant. When dry sodium carbonate is employed, 1 g. of the material to 5 c. c. of water makes a practically concentrated solution; while if the crystals are employed 2.7 g. of the carbonate to 5 c. c. of water will be required. This is due to the fact that the crystals contain water of crystallization. Sodium carbonate will precipitate lithium, iron, manganese, zinc, silver, lead, and cadmium as carbonates, and aluminum, chromium, and copper as hydrates.

METALS

63. Metals as Reagents.—Metal reagents are used for precipitating metals from solutions. This may take place by an interchange of metals or by electrolysis, and probably in some cases by contact action, or *catalysis*. The purer the metals are for reagents, the more reliable will be the reactions and the results.

64. Aluminum.—Aluminum, *Al*, in sheet form is used in copper assays, and in the precipitation of bismuth.

65. Copper.—Copper, *Cu*, is used in electrolysis.

66. Lead.—Lead, *Pb*, in sheet and granulated form is used in the laboratory as a precipitant for copper, and for the reduction of iron from the ferro to the ferric state.

67. Iron.—Iron, *Fe*, in the form of wire is used for standardizing and reducing purposes.

68. Tin.—Tin, *Sn*, is used for precipitation and for indicating the presence of certain elements in solution. It is

also an oxidizing agent, and is used for reducing some metallic solutions from the *-ous* to the *-ic* state.

69. Zinc.—Zinc, *Zn*, is obtained in granulated and sheet forms, as well as in filiform threads and as fume. It will precipitate gold and silver from cyanide solutions, and will precipitate lead, copper, arsenic, antimony, iron, etc. from solutions containing those elements. The zinc must be free from arsenic, and contain but a trace of iron. It will reduce ferric compounds to the ferrous state.

70. Normal Acid Solutions.—Normal acid solutions are standard solutions so prepared that each liter must contain the chemical equivalent weighed in grams, of the hydrogen-replacing element or group of elements. For example, normal acid solutions are prepared so that 1 liter of dilute acid shall contain 1 g. of replaceable hydrogen. Taking the molecular weight of sulphuric acid and assuming that it stands for grams, then $H_2 + S + O_4 = 2 + 31.82 + 63.52 = 97.34$ g. This weighed quantity placed in a liter of water would represent a standard solution having a strength of $\frac{97.34}{1,000}$ or .0973 g. per cubic centimeter; but as there are two replaceable atoms of hydrogen in sulphuric acid, the chemical equivalent is $.0973 \div 2 = .04865$ g. per cubic centimeter in a normal sulphuric-acid solution.

Normal hydrochloric-acid solutions are calculated as follows: $H + Cl = 1 + 35.18 = 36.18$ g. H_2SO_4 per liter; hence, 1 c. c. will contain .036 g. of HCl.

Normal nitric-acid solutions contain $H + N + O_3 = 1 + 13.93 + 47.64 = 62.57$ g. HCl per liter, and 1 c. c. contains .062 g. of HNO_3.

Normal oxalic acid contains $C_2 + O_4 + H_2 = 23.82 + 63.52 + 2 = 89.34 \div 2 = 44.67$ g. HNO_3 per liter, and 1 c. c. contains .044 g. of $C_2O_4H_2$. When the formula $C_2O_4H_2 + 2H_2O$ is taken, 1 c. c. contains .062 g. of oxalic acid.

ALKALI SOLUTIONS

71. Normal Alkali Solutions.—Normal alkali solutions are those which have, in each liter of alkali solution, the chemical equivalent, weighed in grams, of the hydrogen-replacing element or group of elements. The method of calculating such solutions is as follows:

Normal sodium-hydrate solutions must contain $Na + O + H = 22.88 + 15.88 + 1 = 39.76$ g. of $NaOH$ per liter; hence, 1 c. c. will contain .039 g. of $NaOH$.

Normal barium-hydrate solutions would contain $Ba + O_2 + H_2 = 136.4 + 31.76 + 2 = 170.16$ g. of BaO_2H_2 per liter, were it not that there are two replaceable atoms of hydrogen; hence, there will be $\frac{170.16}{2} = 85.08$ g. of BaO_2H_2 per liter, and each cubic centimeter will contain .085 g. of BaO_2H_2.

Normal ammonium-hydrate solutions, $(NH_4)OH$, contains 34.81 g. of $(NH_4)OH$ per liter, and 1 c. c. contains .034 g. of $(NH_4)OH$.

Normal potassium-hydrate solutions, KOH, contain 56 g. of KOH per liter, and each cubic centimeter contains .055 g. of KOH.

Normal sodium-carbonate solutions contain 52.6 g. of Na_2CO_3 per liter, and 1 c. c. contains .0526 g. of Na_2CO_3. Sodium has the same valence as hydrogen, and in this case the valence of oxygen is considered thus: $Na_2O + CO_2 = Na_2CO_3$. $Na_2CO_3 = 105.2$, and, since the valence of oxygen is 2, the solution will contain $105.2 \div 2 = 52.6$ g. per liter.

Normal potassium-carbonate solutions contain $K_2 + C + O_3 = 77.7 + 11.91 + 47.64 = 137.25$, and $137.25 \div 2 = 68.62$ g. of K_2CO_3 per liter; therefore, 1 c. c. contains .068 g. of K_2CO_3.

72. Half-Normal Solutions.—Half-normal solutions, $N/2$, are one-half the strength of normal solutions, and to prepare them just one-half the number of grams are used per liter.

One half-normal solution of ammonia is found as follows: $NH_3 + H_2O = 34.81$. A normal solution of NH_3 is $\frac{34.81}{2}$

$= 17.4$, and a half-normal solution is $\frac{17.4}{2} = 8.7$ g. per liter; hence, 1 c. c. $= .0087$ g. of NH_3.

73. One-Fifth Solutions.—One-fifth normal solutions are written $N/5$, and are one-fifth the strength of normal solutions; for example, an $N/5$ solution of $K_2Mn_2O_8$ contains 6.25 g. of $K_2Mn_2O_8$ per liter, and is found as follows: $K_2 + Mn_2 + O_8 = 312.94$ g., and $\frac{312.94}{5 \times 10} = 6.25$ g. per liter.

74. Decinormal Solutions.—Decinormal solutions are written $N/10$, and are frequently used where the number of grams of salt in a liter would require strong solutions of the element sought for saturation. As an example, an $N/10$ solution of potassium permanganate contains 3.129 g. of $K_2Mn_2O_8$ per liter. Since 1 c. c. is to saturate 10 c. c. of the element sought, 31.29 g. per liter is required for a normal solution, and 3.12 g. per liter for an $N/10$ solution, and 1 c. c. will contain .003 g. of $K_2Mn_2O_8$.

Potassium bichromate in $N/10$ solutions contains $K_2 + Cr_2 + O_7 = \frac{292.26}{10 \times 10} = 2.92$ g. per liter.

75. Miscellaneous Titrating Solutions.—Where a solution used in titrating depends on some one element for its activity and usefulness, the solutions are made normal for that element; thus, potassium permanganate and potassium bichromate are used on account of their oxidizing powers. Such solutions are frequently made normal to their oxygen; for example, a normal solution of potassium bichromate is $K_2O + Cr_2O_6 = 292.26$. As there are six replaceable atoms of oxygen, $\frac{292.26}{6} = 48.71$ g. of $K_2Cr_2O_7$ per liter are required; hence, 1 c. c. contains .048 g. of $K_2Cr_2O_7$ reckoned in terms of oxygen. In the case of potassium permanganate, a normal solution contains 31.29 g. of $K_2Mn_2O_8$ reckoned in terms of hydrogen; when reckoned in terms of oxygen, each liter will contain $\frac{31.29}{4} = 7.82$ g. of

oxygen, and each cubic centimeter will contain .0078 g. of oxygen.

76. Iodine Solutions.—Iodine solutions are made normal and titrated against other solutions, in order to obtain a factor for calculating the quantity of a third element in solution. A normal solution of iodine contains 126.01 g. of I per liter, and 1 c. c. of this is equivalent to .1260 g. of I. A normal solution of sodium thiosulphate, $Na_2S_2O_3 \cdot 5H_2O$, contains 246.44 g. per liter, and 1 c. c. = .0.246 g. per cubic centimeter. One c. c. of the standard iodine solution is equivalent to 1 c. c. of the standard thiosulphate solution; hence, .126 g. of I is equivalent to .246 g. of $Na_2S_2O_3 \cdot 5H_2O$.

77. Silver-Nitrate Solutions, $AgNO_3 + H_2O$.—A normal solution of silver nitrate contains 168.6 g. of $AgNO_3$ per liter or .1686 g. per cubic centimeter. Usually, $N/10$ solutions are used for titration. According to the equation

$$AgNO_3 + 2KCN = AgK(CN)_2 + KNO_3$$

it requires 2 molcules, KCN, to saturate 1 molecule of $AgNO_3$. A standard silver-nitrate solution can be made up from the molecular weights as follows:

$$\left.\begin{matrix} AgNO_3 \\ 168.68 \end{matrix}\right\} \text{saturates} \left\{\begin{matrix} 2KCN \\ 129.38 \end{matrix}\right.$$

If 168.68 g. of silver nitrate is dissolved in 1,000 c. c. of water, 1 c. c. of the solution will be equivalent to .129 g. of KCN. A decinormal solution would be equal to .013 g. of KCN per cubic centimeter of $AgNO_3$.

A standard silver-nitrate solution can be made in terms of sodium chloride from the equation:

$$\frac{AgNO_3}{168.68} + \frac{NaCl}{56.06} = AgCl + NaNO_3$$

that is, 1 molecule of silver nitrate, $AgNO_3$, saturates 1 molecule of sodium chloride, $NaCl$. Then, if 168.68 g. of $AgNO_3$ is dissolved in 1,000 c. c. of water, each cubic centimeter is equivalent to .056 g. of $NaCl$, or to .035 g. of chloride

STANDARDIZING SOLUTIONS

78. Normal Salt Solutions.—In practice, it is customary to determine the strength of a solution and so standardize it, rather that take a theoretical standard solution. The method followed in determining the strength of a solution is called **standardizing** the solution. The value of titrating one solution against another in order to ascertain its strength becomes evident when it is known that absolutely pure chemicals cannot be obtained, and further that they deteriorate with age. When the strength of a solution has been ascertained by experiment, it is termed a *standard solution* and is used to ascertain the quantity of an element in another solution. It is possible to make up standard solutions so that 100 c. c. of the solution will equal 1 g. of the element sought; this divided by 10 will give the percentage of the element in solution. Such solutions prevent mistakes in calculating percentages, since the burette reading is practically all that is required.

79. Titrating Solutions.—When a standard solution is added to another solution little by little from a burette, it will gradually neutralize that solution, and then precipitate the element sought. Solutions for titration are made up from salts known to react on the substances sought when in solution. In order to know just when the standard solution from the burette saturates the solution being tested, some change of color must take place between the solutions, or, if this is not possible, an indicator that will cause a change of color must be used. Some indicators may be added to the solution being titrated, while in other cases they must be placed in the cavities of a spot plate.

The *end point* is that point where just sufficient of the standard solution has been used to neutralize the solution being tested. Any quantity above or below this point will cause an error in calculations. When nearing the end point, great care must be taken in manipulation. While theoretical solutions are not used as standards, the theoretical weights derived from chemical equations are approximately used.

80. Standard Potassium-Permanganate Solutions. In the determination of iron by **Marguerite's method,** the reaction that occurs depends on the oxidizing properties of potassium permanganate, $K_2Mn_2O_8$. According to the equation

$$10FeSO_4 + 8H_2SO_4 + K_2Mn_2O_8$$
$$= 5Fe_2(SO_4)_3 + 2MnSO_4 + K_2SO_4 + 8H_2O$$

ferrous sulphate is oxidized to ferric sulphate. To accomplish this reaction requires 10 parts of iron to each part of potassium permanganate; hence, $\frac{K_2Mn_2O_8}{10 \times 55.5} = \frac{312.9}{555} = .563$ g. of $K_2M_2O_8$ is required to oxidize 1 g. of iron. To oxidize 10 g. of iron will require 5.63 g. of $K_2Mn_2O_8$ in 1 liter of water; each cubic centimeter will then equal, theoretically, .01 g. of iron, or 1 per cent. of iron in a 1-g. sample.

To prepare an approximately normal solution of potassium permanganate, 5.64 g. of pure crystallized permanganate is dissolved in 1,000 c. c. of distilled water. The solution is placed in a glass-stoppered bottle and shaken from time to time until ready for use. The solution should be made up at least 48 hours before standardizing.

81. Standardizing Potassium Permanganate.—To standardize the solutions, a solution containing a known weight of iron is titrated with the standard solution whose strength is to be determined. The weight of iron in the solution divided by the number of cubic centimeters of the standard solution used, up to the end point, gives the weight of iron that each cubic centimeter of the solution will oxidize. The iron used is in the form of piano wire, which contains 99.7 per cent. of pure iron. The wire should be well rubbed with fine sandpaper or emery paper before weighing out, to remove dirt and the shellac with which it is sometimes covered, to prevent rusting. The pieces for weighing are cut off and coiled around a lead pencil in order to get them into convenient shape for weighing, which is very carefully done on the button balance.

Two separate iron solutions are always run for standardizing, and the average result of the two (if they check within

reasonable limits) is accepted as the standard of the solution. (If they do not check properly, the work must be repeated.) The charges of wire are intentionally made to differ by from 20 to 50 mg. in weight; this makes accurate work necessary in order to get good checks; whereas, if the two charges were of very nearly the same weight, there might be a perfectly unintentional and unconscious "juggling" of results, to make them agree whether they will or not.

The methods of standardizing both solutions are given in the articles that follow.

82. Dissolving the Iron.—Two portions of piano wire of about 200 and 250 mg. each, respectively, are first accurately weighed. Each is then placed in a 250-c. c. flask or beaker and 10 c. c. of *dilute* sulphuric acid (concentrated H_2SO_4 will not dissolve iron) and 10 drops of hydrochloric acid are added.* The whole is then heated gently until the iron is completely dissolved, more H_2SO_4 being added and another drop of HCl if necessary. The solution will take only a few minutes. As soon as the iron is all dissolved, the contents of the flask are diluted to about 200 c. c. with distilled water.

83. Reducing Ferric to Ferrous Salts.—More or less of the iron will be oxidized to ferric sulphate during the solution, and this must be reduced to *ferrous* sulphate before titrating. To reduce the solution, 2 or 3 g. of pure granulated zinc is added, and the solution allowed to stand for a short time. The hydrogen liberated by the action of the acid on the zinc reduces the ferric sulphate to ferrous sulphate. The solution, which is at first tinged yellowish by

*The HCl is added merely in order to have the conditions in the standardization as nearly as possible the same as the conditions in regular determinations, in which it is necessary to use some HCl in dissolving the ore. HCl has a tendency, if there is much of it present in the solution—and particularly if the solution is warm—to decompose the permanganate and cause a high result. By using, however, as little excess of HCl as possible in dissolving the ore and then diluting the solution up largely, adding a considerable excess of H_2SO_4 and titrating the solution cold, the effect of the HCl can be completely counteracted.

the ferric sulphate, soon becomes perfectly colorless. Very small amounts of ferric salts do not color the solutions perceptibly; hence, to be absolutely certain that all the iron is reduced, the solution should be tested with a weak solution of potassium sulphocyanate, $KCNS$, or of the corresponding sodium or ammonium salt. A few drops of sulphocyanate solution are put into the depressions of the spot plate, and a drop of the iron solution is taken out on the end of a stirring rod and added to the sulphocyanate on the spot plate. Ferrous salts do not affect the color of the sulphocyanates, so that if the iron is completely reduced there will be no reaction; if there is the *least trace* of ferric salt present in the solution, however, a drop of the iron solution added to the sulphocyanate on the spot plate causes a strong and characteristic red coloration. If, then, the test gives a red coloration, the reduction is incomplete, and must be continued until the sulphocyanate no longer gives any reaction on the addition of the iron solution.

As soon as the reduction is complete, and the excess of zinc has entirely dissolved, the titration should be proceeded with immediately; if allowed to stand very long exposed to the air, some of the iron will reoxidize. The contents of the flasks are transferred to No. 5 beakers, the flasks being rinsed out well with distilled water, and 20 c. c. of dilute H_2SO_4, or a correspondingly smaller quantity of concentrated H_2SO_4 added, which should be poured in slowly and stirred constantly to prevent spurting. Concentrated acid has the disadvantage that it heats the solution considerably. (The excess of H_2SO_4 is necessary in the solution, both to promote the desired reactions and to counteract the HCl.) The solution is then diluted with distilled water up to about 700 c. c. in bulk, when it is ready for titration.

84. Titrating Iron With Potassium Permanganate.—In titrating iron with potassium permanganate, the burette is filled exactly to the zero point with the standard solution. The burette should always be rinsed out with distilled water before using, and then a few cubic centimeters

of the standard solution run through it and thrown away before filling it with the solution. When everything is ready, a little of the iron solution should be poured into a small beaker, to hold in reserve, and the main portion titrated, either in the beaker or in a titrating dish. At first, several cubic centimeters of the standard solution may be run in at a time, stirring briskly all the while. As the end point approaches, more caution should be observed, a few drops only at a time being added. As the permanganate strikes the iron solution it becomes first brown and then colorless. The action becomes slower and the brown less intense toward the end point, and, as soon as the end point is passed, the permanganate no longer breaks up and decolorizes on entering the iron solution, but retains its purple color, tingeing the iron solution a faint pink. The end point may be safely passed in the first titration, as there is more than enough solution in reserve to bring it back. This reserve is now added, the beaker being rinsed out thoroughly with distilled water, and the titration finished very carefully, drop by drop. The end point is reached when a single drop of the standard solution added to the iron solution causes a faint, permanent pink tinge (a pink tinge lasting 1 minute may be considered as "permanent" for purposes of comparison). Every chemist has his own particular end point, but whatever tinge is adopted for the end point when standardizing, the same tinge must be used as the end point of all titrations with that solution. The paler the tinge accepted as the end point, the more nearly exact will be the result of the analysis, as the *actual* end point is not the point at which the color shows, but is one drop or portion of a drop short of that point—when all the iron is oxidized, but there is no free permanganate in the solution.

85. The other solution is titrated in the same way, and the results of the two titrations are calculated and averaged to obtain the standard of the permanganate solution. The following examples will illustrate the method of calculation better than a long verbal explanation: Suppose that the two

charges of iron wire weighed .2054 g. and .2396 g., respectively, and that the titrations consume 19.9 c. c. and 23.3 c. c. of permanganate solution, respectively. The iron wire is about 99.7 per cent. pure iron, and the standard is equal to $\frac{\text{grams taken} \times 99.7}{\text{no. c. c. of } K_2Mn_2O_8 \text{ used}}$; then, there is $.2054 \times .997 = .20478$ g. of pure iron in the first solution, and $.2396 \times .997 = .23888$ g. of pure iron in the second solution.

In the titration of the first solution, therefore, 19.9 c. c. of permanganate solution oxidizes .20478 g. of iron, or each cubic centimeter of permanganate solution oxidizes $\frac{.20478}{19.9} = .01029$ g. (or 10.29 mg.).

In the titration of the second solution, 23.3 c. c. of permanganate solution oxidizes .23888 g. of iron, or each cubic centimeter oxidizes $\frac{.23888}{23.3} = .01025$ g. (or 10.25 mg.).

The average of these two results is the standard of the solution. Thus, $\frac{.01029 + .01025}{2} = .01027$ g., or the amount of iron each cubic centimeter of the standard solution will oxidize.

The bottle containing the solution is then labeled with the name or formula ($K_2Mn_2O_8$) of the solution and the standard; thus:

Potassium Permanganate
1 c. c. = .01027 g. of *Fe*

86. The solution should be kept in a cool, dark place when not in use, as it slowly decomposes and loses strength if exposed to the light. It should be restandardized every few weeks, as the strength changes slightly with time. If it is restandardized 2 or 3 weeks after the first standardization, the results will show how fast it is changing strength, and from this the chemist will know about how often it will be necessary to restandardize the solution. (A closet under the table or sink is very convenient for standard solutions and other chemicals that are sensitive to the light.)

87. Standard Bichromate Solutions.—In the presence of free acid, potassium bichromate will convert a ferrous oxide into a ferric oxide, according to the formula

$$6FeCl_2 + K_2Cr_2O_7 + 14HCl = 3Fe_2Cl_6 + 2KCl + Cr_2Cl_6 + 7H_2O$$

This is termed *Penny's method*, and the formula shows that 1 part of bichromate solution will convert 6 parts of iron from the ferrous state to the ferric state. Thus, $\frac{K_2Cr_2O_7}{6Fe} = \frac{292.26}{333} = .877$ g. of $K_2Cr_2O_7$ will satisfy 1 g. of iron. To oxidize 10 g. of iron will require 8.77 g. of $K_2Cr_2O_7$ in 1 liter of water, and each cubic centimeter will equal theoretically .01 g. of iron.

88. The standard bichromate solution is usually made approximately a half-normal solution; that is, 1 c. c. of the solution will oxidize about 5 mg. of iron. The end point in this method is very sharp, and there is less danger of running beyond it with a weak solution than with a strong one, in which each drop contains two or three times as much bichromate as a drop of the weak solution. Some chemists recommend using a strong solution (normal or even somewhat stronger) until nearly to the end point, and then finishing the titration with a decinormal ($\frac{1}{10}$-normal) solution (1 c. c. of solution = 1 mg. of iron). The half-normal solution is dilute enough, however, and its use obviates the necessity for making up two solutions and of taking two separate readings of the burette for each titration and then figuring up the amount of iron oxidized by each solution. The half-normal solution is prepared by dissolving 4.39 (or, roughly, 4.4) g. of pure potassium bichromate in 1 liter of distilled water. The solution should be allowed to stand for a day or two before standardizing.

89. Reducing Ferrous Chlorides to Ferric Chlorides.—For titration with potassium bichromate, the iron may be in solution as either ferrous sulphate or ferrous chloride. Two charges of piano wire, of between 100 and 200 mg. each, are weighed up carefully, placed in 250 c. c.

flasks or beakers, and dissolved by boiling in either dilute sulphuric acid or dilute hydrochloric acid (5 c. c. concentrated HCl and 20 c. c. distilled water). The solutions are diluted up to about 200 c. c. each, and the ferric iron is reduced to the ferrous state; then, the solutions are transferred to large beakers, diluted up to about 500 c. c. each, and titrated. The method of titration is the same whether hydrochloric or sulphuric acid is used in the solution. If the iron is in the ferric state, it must be reduced to the ferrous state either by means of zinc or by introducing several grams of granulated lead into the solution and boiling until the reduction is complete. Ferric chloride in solution may be reduced by either of these methods, or else, much more quickly, by means of a moderately strong solution of stannous chloride ($SnCl_2$ = bichloride of tin).

The reduction with granulated lead is accomplished as follows: The solution is first heated nearly to boiling over a Bunsen burner, and then about 5 g. of test lead is added. The solution is then boiled for some time. The yellow tinge gradually fades and the solution finally becomes perfectly clear. At this point 5 g. more of test lead is added. The solution is tested from time to time with potassium sulphocyanate. As soon as the solution is completely reduced and no longer gives a red coloration or ferric reaction with the sulphocyanate, it is poured off from the lead into a large beaker. The lead is washed several times, the washings being added to the main solution. The solution is then diluted and titrated.

90. Reduction of Ferric Chloride by Stannous Chloride.—The reduction of ferric chloride by means of stannous chloride is quick and simple. The solution is warmed and then a dilute solution of stannous chloride is added, drop by drop, stirring after each drop, until the iron solution becomes colorless. A few drops are usually sufficient. After the solution has become perfectly clear and colorless, one more drop of stannous chloride is added to make complete reduction certain (or the solution is tested

with sulphocyanate to see that all the iron is reduced). The slight excess of stannous chloride must then be oxidized by the addition of a large excess of mercuric chloride. About 20 c. c. of a saturated solution of mercuric chloride is added, *all at once*—and immediately after the final drop of stannous chloride—to the iron solution, which should be stirred rapidly to distribute the mercuric chloride quickly throughout the entire solution. The mercuric chloride instantly oxidizes the stannous chloride to stannic chloride, and is itself reduced, forming a dense, curdy, white precipitate of mercurous chloride. This precipitate does not interfere in any way with the subsequent reactions. The mercuric chloride *must*, however, be added suddenly and in great excess; if it is not in considerable excess or is added slowly—which, as the reaction between the stannous chloride and the mercuric chloride is almost instantaneous, has the same effect as adding it in small quantity—the stannous chloride will reduce part or all of it to gray, metallic mercury, and not only is it then impossible to tell with absolute certainty whether the stannous chloride is completely oxidized, but the mercury also renders the result of the titration unreliable. If the precipitate is perfectly white, the chemist is absolutely certain that the oxidation of the stannous chloride is complete, and that his results, if the rest of the work has been done carefully, are correct; but if there is even the faintest tinge of gray, there is room for doubt, both as to the completeness of the oxidation and as to the accuracy of the work.

91. Titrating Iron With Bichromate Solution. As soon as the reduction is complete, the solution is diluted up to about 500 c. c. with distilled water, and about 5 c. c. of strong HCl is added. A reserve portion is poured off, and the main solution is then titrated with the bichromate solution, which is run in from the burette. The solution, which was colorless at first—or white, if it was reduced by stannous chloride—soon acquires a pale green tint, which becomes darker as more bichromate is added and stirred in. Should it turn brown, more HCl should be

added. After the color becomes dark green, the bichromate should be added more carefully, and the iron solution should be tested after each addition by adding a drop of it to an indicator solution of potassium ferricyanide on the spot plate. (The ferricyanide solution should not be too strong and should be free from *ferrocyanide.*) Ferricyanide gives a blue precipitate with solutions of ferrous salts, even in the most minute quantities, but is not affected by ferric salts; consequently, the indicator solution will turn deep blue with the first additions of the iron solution, but the blue coloration will become paler and paler as the titration proceeds and the bichromate solution oxidizes more and more of the iron, and when the last trace of the ferrous iron is oxidized to the ferric state, it will cease entirely. The point at which the blue coloration ceases, therefore, marks the end of the titration. The end point may be passed in titrating the main solution, as in the permanganate method; then, the reserve solution should be added, and the beaker rinsed out carefully with distilled water (the rinsings being added to the main solution); the solution is now titrated very carefully, drop by drop, to the final end point. The second solution may be titrated in the same way. The solutions may be titrated either warm or cold; they should, however, both be titrated at about the same temperature, and the temperature adopted in the standardization of a solution should be retained in all subsequent determinations with that solution.

92. Calculating Bichromate Titrations.—The results in calculating bichromate titrations are obtained exactly as in the permanganate standardization: the weight of iron in the solution divided by the number of cubic centimeters of standard solution used equals the strength, or standard, of each cubic centimeter of the standard solution. The average of the two determinations is taken as the standard of the solution.

EXAMPLE 1.—Two charges of piano wire weighing, respectively, 150 mg and 205 mg. were dissolved in HCl, diluted to 500 c. c., and titrated with a bichromate-of-potassium solution. It was found that

it required 14 c. c. and 19 c. c. of the solution to saturate the iron solution. What was the standard strength of the potassium-bichromate solution?

SOLUTION.—The iron wire was 99.7 per cent. pure; hence, the solution will saturate

$$\frac{.150 \times .997}{14} = .01068 \text{ g. of } Fe; \ \frac{.205 \times .997}{19} = .01075 \text{ g. of } Fe.;$$

hence, the standard will be

$$\frac{.01068 + .01075}{2} = .01071 \text{ g. of } Fe. \quad \text{Ans.}$$

EXAMPLE 2.—Fifty c. c. of an iron-ore solution containing 10 g. of ore required 56.34 c. c. of the standard permanganate solution to saturate it. One c. c. of the $K_2Mn_2O_8$ corresponded to .01027 g. of iron. (*a*) What is the percentage of metallic iron in the solution? (*b*) What is the percentage of ferric oxide in the solution?

SOLUTION.—(*a*) 56.34 × .01027 = .57861 g. of iron in 50 c. c. of iron solution; then,

$$\frac{.57861 \times 10 \times 100}{10} = 57.861 \text{ per cent. of } Fe. \quad \text{Ans.}$$

(*b*) $\frac{Fe_2O_3}{Fe_2} = \frac{158.64}{111} = 1.429$, and 57.861 × 1.429 = 82.6 per cent. of Fe_2O_3. Ans.

93. Standard Molybdate Solution.—In Alexander's method of lead analysis, ammonium-molybdate solutions are used for titration. This method is based on the reaction that occurs when hot solutions of lead acetate are mixed with ammonium-molybdate solutions. A standard solution of ammonium molybdate is prepared by dissolving 8.6 g. of ammonium molybdate in 1,000 c. c. of water.

Ammonium molybdate has the formula $(NH_4)_6Mo_7O_{24}\cdot 4H_2O$ and a molecular weight of 1,277.32. The molybdenum that unites with the lead to form $PbMoO_4$ is derived from this salt; hence, $\frac{1,237.32}{MnO_3(142.94)} = 8.6$ g. of molybdate salt for 1 g. of $PbMoO_4$, and each cubic centimeter of this solution will contain .01 g. of lead. If the solution is not clear, a few drops of ammonia should be added. As an indicator for this solution, 1 part of tannin in 300 parts of water is used. The tannin solution is placed in drops on a spot plate, and when an excess of molybdate solution is added to the lead

solution, the tannin solution turns yellow. The excess necessary to effect the indicator must be determined, and subtracted from the burette reading. The excess necessary is ascertained by making a preliminary test between the tannin and the molybdate solution.

94. Standardizing Molybdate Solutions.—Two methods are in vogue, when making up standard molybdate solutions: one is to use pure, dry, lead sulphate, and the other is to use pure lead foil. The two will be given here.

1. To standardize an ammonium-molybdate solution, 300 mg. of pure, dry, lead sulphate is dissolved in hot ammonium acetate. This solution is then acidified with acetic acid and diluted with water to 250 c. c. The solution is now boiled and then titrated with the molybdate solution until all the lead is precipitated as $PbMoO_4$. The ammonium molybdate is run into the lead-sulphate solution, which is stirred continually. After each addition of molybdate solution, the liquid should be tested by placing a drop on the tannic-acid solution on the spot plate. When the tannic acid gives a yellow color, the titration is finished.

2. Two pieces of lead foil weighing 300 mg. and 500 mg.. respectively, are dissolved in from 10 c. c. to 15 c. c. of 1-to-1 nitric acid. When the lead is all dissolved, 2 c. c. of 1-to-1 sulphuric acid is added. The whole is then stirred thoroughly and allowed to settle. This is now decanted on the filter paper and washed by decantation three or four times with water containing 2 per cent. of sulphuric acid, the decanting always being done as closely as possible. The beaker is washed out once with a little cold water, as much of the precipitate as possible being kept in the beaker. The lead sulphate that is on the filter paper is now dissolved by pouring over it 50 c. c. of hot ammonium-acetate solution. This solution is passed through the filter a second or a third time if necessary, and the paper washed with hot water. The hot ammonium acetate is then poured over the main bulk of the precipitate. This is heated until it is dissolved, diluted to 200 c. c., made barely acid with acetic acid, and titrated

The ammonium molybdate is run in from the burette with constant stirring, and tested from time to time by placing a drop of the solution on a drop of the tannic-acid solution on the spot plate. When the tannin gives a yellow color, the titration is finished.

95. Calculation of Molybdate Solutions.—The special purpose of a molybdate solution is to precipitate the lead in lead-sulphate solutions. To ascertain the quantity of lead in such a solution, the weight of the salt taken is multiplied by the factor for lead in lead sulphate; thus, $\frac{Pb}{PbSO_4} = \frac{205.35}{300.69}$ $= .68292$; and 300 mg. $\times$ .68292 = .20487 g., or the amount of lead in the solution. Theoretically, the standard solution should precipitate .01 g. of lead per cubic centimeter; hence, it would take 20.48 c. c. to precipitate the amount of lead just mentioned.

EXAMPLE 1.—Two charges of pure lead sulphate, one weighing 250 mg. and the other 275 mg., were used to standardize a solution of ammonium molybdate. The first charge required 26.5 c. c., and the second charge 28 c. c. What was the standard of the solution?

SOLUTION.— $\frac{.250 \times .68292}{26.5} = .00644$, and $\frac{.275 \times .68292}{28} = .00670$; whence,

$$\frac{.00644 + .00670}{2} = .00657. \text{ Ans.}$$

EXAMPLE 2.—Two pieces of pure lead foil weighing 325 mg. and 350 mg., respectively, were dissolved in nitric acid and converted into lead sulphate by sulphuric acid. (*a*) What weight of lead sulphate was made in each solution? (*b*) What was the standard of the molybdate solution when it required 32.4 c. c. in one case, and 34.6 c. c. in the other to neutralize the lead-ammonium-acetate solution?

SOLUTION.—(*a*) The percentage of SO_4 in $PbSO_4$ is 31.707, and the factor is .31707. .325 $\times$ .31707 = .103, and .103 + .325 = 448 mg. .350 $\times$ .31707 = 111, and .111 + .350 = 461 mg. Ans.

(*b*) $\frac{.325 \times .68292}{32.4} = .00685$, and $\frac{.350 \times .68292}{34.6} = .00690$; whence

$$\frac{.00685 + .00690}{2} = .00687. \text{ Ans.}$$

96. Standard Potassium-Cyanide Solutions.—Standard potassium-cyanide solutions, $KCN + H_2O$, are used in

the volumetric determination of copper. If a potassium-cyanide solution is added in excess to a blue ammonical solution of copper nitrate, the latter will be decolorized. The reaction that occurs is such as to form a double salt of copper potassium cyanide, which, according to Remsen, has the formula $KCu(CN)_2$, or $K_2Cu(CN)_4$. Remsen states that, under ordinary circumstances, a compound having the formula $K_2Cu(CN)_4$ is formed. For the purposes of calculation, the reactions represented by the following equations may be used:

$$2Cu + 4KCN + O + H_2O = 2CuK(CN)_2 + 2KOH$$
$$Cu(NO_3)_2 + 2KCN = Cu(CN)_2 + 2KNO_3$$

According to the first equation, $\frac{4KCN}{2Cu} = \frac{258.76}{126.2} = 2.05$; that is, 1 g. of copper will saturate 1.025 g. of KCN; and according to the second equation, $\frac{129.38}{124.67} = 1.025$, or 1 g. of copper will saturate 1.025 g. of KCN.

As it is difficult to obtain chemically pure potassium cyanide, the solutions are made strong, and then diluted and standardized against pure copper-nitrate solutions.

The cyanide solution should be approximately half normal (1 c. c. = 5 mg. copper). A solution of about this strength may be made by dissolving 22 g. of commercial potassium cyanide (chemically pure cyanide is unnecessary) in 1 liter of distilled water. The solution should be kept in a tightly stoppered colored-glass bottle (dark green glass is best) in a cool, dark place; or, if a dark place is not available, the bottle should be covered with black paper, as the cyanide decomposes quite rapidly under the influence of light. Some chemists also pour in a little coal oil above the cyanide solution in the bottle to further protect it from decomposition. The solution should be restandardized frequently. Great care should be exercised in handling it, as it is extremely poisonous

97. Standardizing Potassium-Cyanide Solutions. The cyanide solution is standardized by titrating solutions containing known weights of copper. Two charges of

chemically pure copper foil, of between 200 and 300 mg. each (the weights of the two charges should vary considerably, as in the case of the iron wire for standardizing the permanganate solution), are weighed and placed in an Erlenmeyer or flat-bottomed glass flask of about 250 c. c. capacity, and 5 c. c. of concentrated HNO_3 added. The copper will immediately dissolve and the flasks will be filled with dense red fumes of nitric oxide. The flasks are now placed on the hot plate and heated until the red fumes are completely expelled. After removing the flasks, the contents of each are diluted to about 100 c. c. with distilled water, and 10 c. c. of strong ammonia water added. Copper hydrate is formed and immediately dissolves in the excess of ammonia, giving a deep-blue solution. The solution is now ready for titration.

The cyanide solution is now run in from a burette (the burette should be rinsed with water and finally with a little of the cyanide solution before starting the titration) until the color begins to fade. The solution is then allowed to stand for about 10 minutes, when it is diluted with distilled water to about 200 c. c. The titration is finished very carefully, the flask being shaken after each addition of cyanide. It is advisable to hold a little of the copper solution in reserve, in case the end point is accidentally passed. The end point most commonly used is the point at which only the faintest tinge of pink shows at the upper edges of the solution when the flask is held against a white background in a good light. Many chemists stop somewhat short of this, while the entire solution retains a pink tinge; and some go beyond, titrating until *all* the color has disappeared. The latter practice is attended with considerable risk of running high, however; with the former it is rather difficult always to strike the same tint, and even if there is no error made in this way, unless the amount of copper in the ore solutions is approximately the same as that in the copper solutions used for standardizing, there will still be a slight discrepancy, as the exact amount of unconverted copper hydrate necessary to impart the pink tint of the end point is not known. The nearer to colorless the titration

is carried, the smaller will be the error due to unconverted copper hydrate.

98. Calculating the Strength of Solution.—To ascertain the quantity of copper that a cubic centimeter of the *KCN* solution will throw down, the solution is titrated against known quantities of pure copper in solution. The result obtained is marked on the bottle of *KCN* solution.

The method of standardizing is illustrated by the following example:

EXAMPLE.—Two charges of pure copper foil weighed, respectively, .2354 g. and .2554 g. The first when titrated required 24 c. c. of the *KCN* solution, and the second required 26 c. c. What is the standard strength of the *KCN* solution, reckoned in grams of copper?

SOLUTION.— $\frac{.2354}{24} = .00980$, and $\frac{.2554}{26} = .00989$; then,

$$\frac{.00980 + .00989}{2} = .00984 \text{ g.}$$

That is, 1 c. c. of the *KCN* solution is equivalent to .00984 g. of copper. Ans.

99. Cyanide Poisoning.—Potassium cyanide taken internally in very small quantities is a deadly poison. It acts almost immediately on the system, affecting particularly the action of the heart; hence, any remedies at hand must be used quickly. The chemical should not be handled with the bare hands, as it is readily soluble and perspiration may cause some of it to adhere. If the hands are thrust into weak solutions of potassium cyanide, it causes, on some persons, bad sores similar to boils. The poisonous gases liberated during the acid treatment of materials containing potassium cyanide are also a cause of poisoning. In some cases, the fumes arising from lixiviation vats where agitation has been in progress have caused nausea, which is the first symptom of cyanide poisoning. Assayers, however, at copper plants make thousands of copper assays yearly without feeling any bad effects.

100. Antidotes for Cyanide Poisoning.—In the case of poisoning caused by placing the hands in cyanide solutions, a salve should be applied that is made up of equal

parts of *ichthyol and boric-acid ointments*. Wetting of the hands should be avoided until the inflammation has been reduced. It will require some time to effect a complete cure.

In the case of poisoning by cyanide fumes, *peroxide of hydrogen*, H_2O_2, is sometimes an antidote. It is injected hypodermically in solutions of from 2½ to 3 per cent. The injections are made every 4 minutes at different parts of the body, while at the same time the stomach is to be washed out with a 2-per-cent. hydrogen-peroxide solution. Peroxide of hydrogen, H_2O_2, forms with hydrocyanic acid, HCN, "oxamide," $CONH_2$, which is a harmless compound; thus: $2HCN + H_2O_2 = 2CONH_2$. In case the patient is unconscious, his mouth must be forced open, and his stomach filled with the solution and then washed out. After this treatment, resort is had to artificial respiration—such as is given in cases of apparent drowning and asphyxiation, or in accidents from electricity.

At the Australian cyanide plants, the following chemicals and apparatus are kept in marked places: a tin box containing a bottle of ferrous-sulphate solution, made up of 7.5 g. of salt and 30 c. c. of water; a bottle containing 1.5 g. of caustic soda dissolved in 300 c. c. of water; and a tube containing 2 g. of magnesia. The chemicals are hermetically sealed, and are kept in the box, which has a cover. When the solutions are used they are mixed in the tin box, and the magnesia is added to the solution. If the patient is conscious, he must drink the antidote at once, followed by half a pint of water. If vomiting is not induced, the patient must be pumped out. In case the patient is unconscious, the solution is poured down a stomach tube and then the stomach pumped out. In this case, a gag will be required, and the operation of washing out the stomach is repeated several times. Artificial respiration is next induced, by the use of ammonia or smelling salts, or, if these do not prove effective, by the adoption of the means employed to resuscitate apparently drowned persons.

101. Standard Potassium-Ferrocyanide Solutions. When a solution containing a zinc salt is brought in contact

with an excess of an aqueous solution of ferrocyanide of potassium, $K_4FeCy_6 + H_2O$, zinc ferrocyanide, $Zn_2Fe(CN)_6$, is precipitated. The precipitate is white, and has the composition $Zn_3Fe_2K_2(CN)_{12}$, as derived from the equation:

$$2K_4Fe(CN)_6 + 3ZnCl_2 = Zn_3Fe_2K_2(CN)_{12} + 6KCl$$

From this equation it requires $\frac{2K_4Fe(CN)_6}{3Zn} = \frac{731.88}{194.7}$ $= 3.81$ g. of potassium ferrocyanide to satisfy 1 g. of zinc. If 1 c. c. of the solution is to be made equal to .01 g. of zinc, then 38.1 g. of the salt is dissolved in 1 liter of water.

102. Standardizing Potassium Ferrocyanide.—To standardize potassium ferrocyanide, two charges of between 200 and 250 mg. each, are weighed out of chemically pure zinc oxide that has previously been heated in a porcelain crucible, to drive off moisture and convert any carbonate of zinc into oxide. The oxide will turn yellow on heating, but will resume its white color when cold. The two accurately weighed charges are then transferred to beakers of about 300 c. c. capacity, and the oxide is dissolved by adding 5 c. c. of concentrated HCl. The whole is then diluted with about 50 c. c of distilled water. Ammonia in slight excess is now added, and then neutralized with HCl, litmus paper being used as an indicator. (The addition of ammonia and its neutralization with HCl are not necessary, but are done simply to have the conditions of the standardization as nearly as possible the same as the conditions of an actual analysis.) To the neutral solution an excess of 10 c. c. of concentrated HCl is now added, the solution diluted to 250 c. c. with cold distilled water, and then titrated, a solution of uranium acetate, $UO_2(C_2H_3O_2)_2$, being used as an indicator. The end color is brown when a saturated solution of uranium acetate is used. The uranium-acetate solution is clarified with a few drops of acetic acid; as the solution decomposes rapidly when exposed to the light and air, it should be kept in a tightly stoppered bottle and in a dark place.

As long as there is not an excess of ferrocyanide, the uranium solution on the spot plate is not affected by the

addition of a drop of the zinc-chloride solution; as soon, however, as there is the least excess of ferrocyanide, the indicator turns brown, thus marking the end of the titration.

In all subsequent titrations, care must be taken to have the conditions as nearly as possible the same as in the standardization; the bulk of the solution should always be the same; the zinc solution should be warm (but not too hot to handle) and should contain the same excess of HCl in all cases, and the temperature of the standard solution should not vary much, if good results are desired. The precipitate of ferrocyanide of zinc should come down pure white, and the solution should be colorless or nearly so.

103. Calculation of Potassium-Ferricyanide Solutions.—The charges of zinc oxide taken for titration were 200 mg. and 250 mg., respectively. These are reduced to their equivalent of metallic zinc as follows: $\frac{Zn}{ZnO} = \frac{64.9}{80.78}$, which gives .80341 as the factor for zinc. $200 \times .80247 = 160.49$ mg. of zinc, and $250 \times .80247 = 200.61$ mg. of zinc. On titrating, it was found that 16.2 c. c. and 20.3 c. c. were required to saturate the zinc solutions; hence, the standard of the solution would be $160.49 \div 16.2 = .0099$ g. of zinc per cubic centimeter, and $200.61 \div 20.3 = .00988$ g. of zinc per cubic centimeter. The standard, therefore, is .00989 g. of zinc per cubic centimeter.

104. Sodium Thiosulphate Solutions.—Potassium iodide will precipitate all the copper from an acetic-acid solution as cuprous iodide, liberating at the same time iodine, according to the equations:

$$\underset{\text{copper acetate}}{2Cu(C_2H_3O_2)_2} + \underset{\text{potassium iodide}}{4KI} = \underset{\text{cuprous iodide}}{Cu_2I_2} + \underset{\text{iodine}}{I_2} + \underset{\text{potassium acetate}}{4KC_2H_3O_2}$$

$$\underset{\text{sodium thiosulphate}}{2Na_2S_2O_3 \cdot 5H_2O} + \underset{\text{iodine}}{I_2} = \underset{\text{sodium iodide}}{2NaI} + \underset{\text{sodium tetrathionate}}{Na_2S_4O_6} + \underset{\text{water}}{5H_2O}$$

From these equations, 1 g. of iodine will saturate 1.96 g. of sodium thiosulphate; hence, the standard solution should

contain 20 g. of thiosulphate in 1 liter of water. The iodine is measured by the thiosulphate, using starch as an indicator, which turns iodine solutions blue.

The starch solution is made by shaking up 1 g. of finely powdered starch in a few cubic centimeters of cold water, thus making a thin paste, after which it is mixed with about 200 c. c. of boiling water. The solution sours, and should be made up fresh every few days. It is used cold.

105. Standardizing Thiosulphate Solutions.—To standardize thiosulphate solutions, two pieces of copper foil, of .2 g. each, are dissolved in 5 c. c. of nitric acid and evaporated to 3 c. c. Five c. c. of hot water and 6 c. c. of ammonia are then added, the solution is boiled a few minutes and then allowed to cool, after which it is diluted to 75 c. c. and 8 c. c. of acetic acid and 5 g. of crystallized potassium iodide added. This is shaken until all the copper has been dissolved and precipitated, and then the free iodine in solution is titrated. To accomplish this, thiosulphate is run in from the burette until the brown color of the iodine changes to yellow; then, from 2 c. c. to 4 c. c. of the starch solution is added, and followed by more thiosulphate until the blue color has disappeared. When near the end point, the solution must be stirred thoroughly and the thiosulphate added, drop by drop.

EXAMPLE.—Two pieces of copper foil weighed .213 g. and .226 g., respectively. These were dissolved and used to ascertain the strength of a thiosulphate solution. It was found that it required 20 c. c. of thiosulphate for one solution, and 22 c. c. of thiosulphate for the other solution, to decolorize the iodine and form sodium iodide. What was the strength of the solution in terms of copper, the molecular weight of iodine being 126.54 and that of copper 63.18?

SOLUTION.— $.213 \div 20 = .01065$, and $.226 \div 22 = .01073$; hence, $\frac{.01065 + .01073}{2} = .01069$ g. of thiosulphate solution is equivalent to .01 g. of iodine. $126.54 : 63.18 = .01069 : x$; hence, $x = .0053$ g. of copper per cubic centimeter of thiosulphate. Ans.

ASSAYING

(PART 5)

VOLUMETRIC AND GRAVIMETRIC ANALYSIS

VOLUMETRIC ANALYSIS

DETERMINATION OF IRON IN ORES

1. Treatment of Iron Ores.—Ordinarily, hematite, limonite, magnetite, and siderite will yield all their iron by boiling with hydrochloric acid. Sulphide ores will not always yield up their iron to hydrochloric acid alone, and are therefore treated with a mixture of nitric, hydrochloric, and sulphuric acids. Occasionally, an ore is encountered that will not yield all its iron, even to the combined action of all three acids; such ores are fused with sodium carbonate, or potassium bisulphate, which makes them soluble. Special methods of dissolving iron in ores are worked out and adopted by assayers, but nearly all of them use either **Penny's method,** which is the bichromate-of-potash-solution method of titration, or else **Marguerite's method,** which is the potassium-permanganate-solution method of titration for determining the iron.

2. Treatment of Oxidized Iron Ores.—Duplicate charges of ore, of 1 g. each—or $\frac{1}{2}$ g. if the ore runs very high in iron—are treated in small casseroles or beakers with 5 c. c. of concentrated HCl. (The use of small vessels in

dissolving ores is always desirable, as it avoids a large excess of acid.) The casseroles or beakers should be covered with watch glasses, concave side up, and should be heated slowly, preferably on a sand bath—a shallow pan of sand set over a burner—to avoid spurting and bumping. The solution usually requires about 30 minutes; when it is complete, the insoluble residue—usually consisting for the most part of silica—is colorless and free from black specks of undecomposed ore. Somewhat refractory ores may sometimes be gotten completely into solution by boiling down to dryness and then adding about 3 c. c. more acid and heating again; or, if the iron is subsequently to be reduced by stannous chloride, a few drops of stannous chloride (2 or 3 are sufficient) added to the acid used in dissolving the ore will aid greatly in getting the ore into solution.

When the ore is completely decomposed, it is diluted with distilled water, and the insoluble residue stirred and rubbed with a rubber-tipped glass stirring rod (a *policeman*), to break up any clots that may have formed. Small specks of solution that have dried on the sides of the vessels and on the watch glasses may be dissolved by rubbing them with a stirring rod moistened with the dilute-acid solution from the casserole or beaker and then washing them off into the main solution with distilled water from the wash bottle.

The solution is then heated nearly to boiling and filtered into the flask for reducing. The heating is not absolutely necessary, but it takes very little time, as the vessel may be heated with only a screen between it and the flame, and hot solutions filter much more rapidly and are cleaner than cold solutions. The vessel and the insoluble residue are washed three or four times with distilled water and the washings run through the filter into the main solution. (In washing the insoluble residue, the solution should be decanted off very carefully on to the filter, leaving as much of the residue in the casserole or beaker as possible until the final washing, as it can be washed much more rapidly and to better advantage in the vessel than in the filter.) The filter paper itself is finally washed by the jet from the wash bottle and the

washings are allowed to run through. A fresh filter paper should always be used for each solution.

The solution is now diluted with distilled water to about 200 c. c.; 5 c. c. of concentrated H_2SO_4 or HCl—according to the method of reduction and titration to be pursued—is added (20 c. c. of dilute H_2SO_4 may be used instead of the 5 c. c. of concentrated acid); and the solution is then reduced by zinc, lead, or stannous chloride in the same way as the solution was reduced for standardizing. See *Assaying*, Part 4.

As soon as the iron is completely reduced, each solution should be diluted to about 700 c. c. if the permanganate method is to be employed for the titration, or to 500 c. c. if the bichromate method is to be used; a further excess of 5 c. c. of acid is added for safety, and then the titration is concluded. The titration is conducted exactly as in the standardization, with the same precautions, the same end point, and, as nearly as possible, the same conditions.

The amount of iron in the solution is determined by multiplying the number of cubic centimeters of standard solution used by the standard of the solution. If 1-g. charges of ore are used, the percentage of iron is given directly; if ½-g. charges are taken, the result must be multiplied by 2 to obtain the percentage of iron in the ore. The average result of the duplicate titrations—which should agree within .1 to .2 per cent.—is taken.

EXAMPLES FOR PRACTICE

1. Duplicate 1-g. ore charges are run. The titration of one solution consumes 37.3 c. c. of the standard solution; of the other, 37.4 c. c. The standard of the solution is .0097 (1 c. c. = .0097 g. of iron). How much iron does the ore contain? Ans. .3623, or 36.23 per cent.

2. Ore charges ½ g. (500 mg.) each. Standard solution used in titration, 27.6 and 27.5 c. c., respectively. Strength of standard solution, 1 c. c. = .0109 g. of iron. How much iron does the ore contain? Ans. .6006, or 60.06 per cent.

3. Fifty c. c. of a 1-g. iron-ore solution reduced with zinc required 34.65 c. c. of standard bichromate solution. One c. c. of $K_2Cr_2O_7$ solution was equivalent to .0168 g. of iron. (*a*) What was the percentage of iron in the ore? (*b*) What was the percentage of ferric oxide?

Ans. { (*a*) 58.21 per cent. of Fe
(*b*) 83.18 per cent. of Fe_2O_3

3. Treatment of Sulphide Iron Ores.—Duplicate charges of .5 g.—or 1 g. if the ore runs low in iron—are dissolved in small casseroles or flasks, with 2 c. c. of strong HCl, 5 c. c. of strong HNO_3, and 8 c. c. of dilute H_2SO_4, added in the order named. The dilute acid should consist of 60 per cent. concentrated H_2SO_4 and of 40 per cent. water. In diluting H_2SO_4, the acid should always be poured gradually into the water—and the mixture stirred constantly while pouring. If the ore is dissolved in flasks, the subsequent reduction may be made in the same flasks. The solution should now be heated on a sand bath or a hot plate until dense white fumes of sulphurous oxide, which are recognized by their color and suffocating sulphurous odor, are evolved. The purpose of the sulphuric acid is merely to make sure that the last trace of nitric acid is expelled. The nitric acid is necessary to break up the sulphides; but if the least trace of it remained in the solution, it would ruin the titration. *The boiling point of H_2SO_4 is so much higher than that of HNO_3 and HCl that these last-named acids are completely evaporated from the solution by the time the H_2SO_4 begins to break up and liberate sulphurous fumes;* hence, the liberation of copious sulphurous fumes is a pretty sure indication that the nitric acid is all expelled from the solution. The heating should be continued for a few minutes longer, to make sure that the last trace of HNO_3 is removed. The solution is then cooled and diluted very carefully with distilled water. No attempt should be made to add water to the solution while it is hot, as the reaction between the water and the hot H_2SO_4—which has been concentrated by evaporation—might cause a serious accident. The solution is now filtered, the residue washed on the filter, the washings being added to the filtrate, and the reduction and titration proceeded with as in the case of oxidized ores. The stannous-chloride reduction cannot be used, on account of the iron being in the form of a sulphate; the reduction with zinc will be found the most satisfactory. This method is applicable to all sulphides—lead, copper, zinc, or iron—in which the percentage of iron is to be determined. Arsenic or antimony

in the solution will cause a high result in the permanganate titration, and should be precipitated by H_2S and filtered off before titrating.

4. Treatment of Refractory Iron Ores.—If an ore will not completely decompose in acids, the insoluble residue is caught on a filter paper. The filter paper and residue are placed in a porcelain crucible and subjected to such heat that the paper ignites and burns to a white ash. To the contents of the crucible, 5 g. of potassium bisulphate, $KHSO_4$, is added, and the mass is fused over a Bunsen burner until the fusion becomes quiet. The crucible should be of such size that the residue and the flux will not fill it more than one-third full, as the bisulphate boils up very quickly and rapidly and is liable to overflow if more than this quantity is used. The heat should be slow at first, to avoid boiling over, and should be increased to a dull red at the finish. The fusion will usually require about 15 minutes. The crucible is then removed from the heat and allowed to cool. When it is sufficiently cool, the fused mass is dissolved out of the crucible with boiling water, the solution and undissolved mass transferred to a beaker, the crucible rinsed into the beaker with distilled water, and the contents of the beaker boiled for a few minutes, to disintegrate thoroughly the fused mass. The solution is then run through a filter into the original filtrate from the insoluble residue, and the whole is reduced by zinc and titrated in the usual manner.

5. Fusion With Soda.—The same result as is obtained in Art. 4 may be had by fusing the insoluble residue with bicarbonate of soda in a platinum crucible. The insoluble residue is filtered out and ignited as before, but in a platinum crucible, and then 5 g. of sodium bicarbonate is added and fused over a blast lamp (a gas burner that is given an artificial blast by means of a blower or foot-bellows) until the fusion is quiet. The crucible is then cooled and the mass dissolved out with very dilute HCl or H_2SO_4, boiled and filtered, and the filtrate added to that from the original

insoluble residue as before. The reduction and titration are performed as usual.

6. Treatment of Iron Ores by Stannous Chloride. Owing to the fact that both of the methods given for working an iron ore require at least one filtration and that they are comparatively slow, the chemists dealing with the iron ores of the Lake Superior region have worked out a method for determining iron that is much quicker and easier than either of the old methods, while with careful work the results obtained are fully as accurate. The preparation of the solutions required for the stannous-chloride method of treating iron ores is given in Arts. **7, 8, 9,** and **10.** The method contains a number of interesting chemical reactions. The ore is dissolved in concentrated hydrochloric acid HCl, but as free HCl, if it were present in the solution at the time of titration, would render it impossible to use the potassium-permanganate solution, the work is carried on as follows: To .5 g. of ore 5 c. c. of **stannous chloride,** $SnCl_2$, and 5 c. c. of concentrated HCl are added. The beaker is covered with a watch glass and the contents boiled until the residue is white and flocculent. In this method, the iron is reduced by means of stannous chloride, and it will be found that a small amount of stannous chloride in the solution will greatly assist in dissolving the ore; hence, a portion of it is added as mentioned in Art. **2.** Some chemists use more hydrochloric acid than the amount given, but as a rule 5 c. c. is enough, and an excess is to be avoided. While the solution in the beaker is still hot, more stannous chloride is added from the burette until the solution is light yellow; the solution is then boiled and stannous chloride added slowly, the solution being shaken and stirred until it is colorless, and about 5 drops in excess added. The watch glass and sides of the beaker are now washed with distilled water and 15 c. c. of mercuric chloride added. The mercuric chloride should be added as quickly as possible and stirred into the solution rapidly, to convert the stannous chloride to stannic chloride. After the solution of mercuric chloride is added and stirred in, there

should be a white, silky-looking precipitate formed, but this will not interfere with the titration. To this, 50 c. c. of manganous-sulphate solution, $MnSO_4 + H^2O$, is next added, followed by 300 c. c. of water. The whole solution is then titrated with a standard permanganate solution. The titration should be done in a large beaker placed on or before a white plate or paper, or, better still, in a white bowl. The titration should be rapid, and as the end reaction approaches, a slight darkening of the solution will be noted. The end reaction is a feeble pink, is of short duration, and hence must be ascertained without delay.

7. Stannous-Chloride Solution.—The stannous-chloride solution is made by taking 85 g. of stannous chloride, $SnCl_2$, and adding 50 c. c. of concentrated HCl and 50 c. c. of water. This mixture is boiled until the solution is clear and then diluted up to 1 liter. Metallic tin is kept in the solution to prevent the formation of oxides.

An excess of the stannous-chloride solution in the iron solution is always to be avoided, and in cases where the ore is low in iron it is sometimes necessary to add less than 5 c. c. of this solution while reducing the ore, or else to use 1 g. of ore and add 5 c. c. of stannous chloride and 10 c. c. of concentrated hydrochloric acid. An ore very low in iron may require only a few drops of the $SnCl_2$ solution to perform the reduction.

8. Mercuric-Chloride Solution.—The mercuric-chloride solution, $HgCl_2$, is made by taking 52 g. of mercuric chloride and dissolving in 1 liter of boiling water. This makes a saturated solution.

9. Manganous-Sulphate Solution.—The manganous-sulphate solution, $MnSO_4$, is made up by dissolving 16 g. of $MnSO_4$ in water, and diluting to 175 c. c. Then, 33 c. c. of phosphoric acid, H_3PO_4 (sp. gr. 1.7), and 32 c. c. of H_2SO_4 (sp. gr. 1.84) are added. The sulphuric acid should be added slowly while the solution is being stirred, to avoid undue heating. When this solution is added to the hydrochloric-acid solution of iron, the manganese sulphate is converted

into manganous chloride, thus avoiding the possible decomposition of the permanganate solution by hydrochloric acid, while the phosphoric acid unites with the ferric iron produced, thereby rendering the solution white and making the end reaction plainer to see.

10. Permanganate Solution.—The permanganate solution may be made up especially to work with ½-g. weights of ore by dissolving 2.9 g. of pure crystallized permanganate in 1 liter of distilled water. In making up this standard solution, it is well to have two bottles or carboys in which the solution is kept, and to use from one while the other is being prepared for use. If a fairly accurate solution is desired, it should be made up and the bottles shaken every day for a week, and then kept in the dark for at least another week before using.

11. Treatment of Iron-Carbonate Ores.—Carbonates of iron contain organic matter, which, if left unchanged in the solution to be titrated, is likely to be acted on by the standard permanganate solution. Such ores may be treated by dissolving the ore in as little concentrated hydrochloric acid as possible, and, when the solution is complete, adding a few crystals of potassium chlorate. This solution is evaporated until the chlorine is expelled, when a very small quantity of acid will still remain. From this point the operations can proceed as before, by adding a small quantity of acid and stannous chloride to reduce the iron. If the organic matter is not destroyed, it renders the slight excess of stannous chloride difficult to adjust, as it imparts to the solution a yellow color resembling ferric chloride.

12. Treatment of Standard Iron Ores.—As a rule, assayers employed in the determination of iron ores do not standardize their solutions against pure metallic iron, but employ an iron ore of known composition. Such standard iron ores—the percentage of iron, phosphorus, silica, and alumina in which is given—can be obtained from chemical-supply houses, and any assayer can easily make up a standard of his own and determine its value by means of the standard

received from some supply house. It is good practice to have a bottle of standard ore at hand near the balances and to weigh out one sample of it each day and run it through with the regular samples to be tested. If the standard sample runs high or low, it shows that the permanganate solution has changed, and hence a slight change will have to be made in the results obtained from all the other samples run in that batch. The running of the sample along with the batch of ores will not require more than from 8 to 10 minutes per day where large numbers of samples are being run, and hence it is not much of a tax on the men doing the work, while the fact that the standard is run with each set of samples serves as a check on all the work.

13. Adjusting the Weight of Ore to the Solution. In laboratories where a great number of iron determinations are made each day, the assayers rarely attempt to make the permanganate solution read directly into per cent., but adjust the weight of the ore taken so that the solution at hand will read into the per cent. desired. As an example, suppose that the laboratory is using a standard iron ore known to contain 57 per cent. of iron, and that this standard is titrated with the permanganate solution made up so that it is supposed to read 1 per cent. of iron for each cubic centimeter of solution used when ½ g. of iron ore is taken. Now, if this solution were employed, and it were found that only 56.5 c. c. of solution were required, it is evident that the solution is stronger than the half-normal solution, and hence the weight of ore that must be taken in order that 1 c. c. of solution would equal 1 per cent. of iron can be found by the following equation:

$$56.5 : 57 = .5 : x$$

$$x = \frac{.5 \times 57}{56.5},$$

$$x = .5044$$

So, in weighing up the samples for analysis, the assayer would first set his balances to weigh .5044 (the tenths of a milligram would probably be obtained by means of the rider), and after the weights had been adjusted, work would proceed

as usual, weighing out samples that would just balance this weight. If the standard run out with the ordinary work for any given day did not show the proper percentage of iron, it would indicate that the strength of the titrating solution had changed, and hence the weight of ore taken for the next work would have to be adjusted to the present new strength of the solution.

An experienced operator can usually make from 90 to 100 iron determinations per day by the method just given.

DETERMINATION OF MANGANESE IN ORES

14. Volhard's Method.—The volumetric determination of manganese by the use of potassium-permanganate solutions was first advanced by Volhard. The method is based on the reaction that takes place when a hot solution of manganous sulphate is titrated with potassium permanganate; thus:

$$3MnSO_4 + K_2Mn_2O_8 + 2H_2O = 5MnO_2 + K_2SO_4 + 2H_2SO_4$$

The same standard solution used for iron can be used for manganese if multiplied by the factor .2951.

$$1K_2Mn_2O_8 = 10Fe = 55.5 \times 10 = 555$$
$$1K_2Mn_2O_8 = 3Mn = 54.6 \times 3 = 163.8$$

The manganese standard is $\frac{163.8}{555} = .2951$ of the iron standard; so that, if 1 c. c. of the permanganate solution will oxidize .01 g. of iron, it will oxidize .002951 g. of manganese. From this it is evident that if the number of cubic centimeters of the solution taken is multiplied by .2951 the product will give the quantity of manganese.

15. Treatment of Manganese Ores.—Two slightly different methods in manipulation may be followed in manganese determination. In the first one described, the entire operation is carried on in the vessel in which the first solution is made. One g. of the ore to be tested is dissolved in from 30 c. c. to 40 c. c. of HCl and a few drops of HNO_3 is added to oxidize any ferric iron present. The solution should be effected in a beaker at least 5 inches high by

3¼ inches in diameter at the top and having no lip. When practically all the nitric acid has been expelled, the solution should be allowed to cool somewhat, 50 c. c. of water added, and the whole boiled to make sure that all the salts are in solution. The beaker is then removed from the hot plate and filled with boiling water to within 1 inch of the top. Dry zinc oxide, *ZnO*, to the amount of 1 or 2 g. in excess, is then added, to precipitate the iron as ferric hydrate, which, on settling, will leave the solution above it colorless. The solution will contain the manganese as a chloride. The zinc oxide should be added slowly until all the iron has been precipitated and the acid all neutralized. (For neutralization test, with litmus paper.) The solution should be stirred thoroughly with a glass rod while the zinc oxide is being added. The solution is then ready for titration with potassium permanganate, and it should be vigorously stirred after each addition of permanganate. The oxidation is complete when the solution, after settling, shows a faint pink color. The heavy precipitate of iron hydrate and zinc oxide has the advantage that it carries the precipitated oxide of manganese to the bottom very quickly, thus enabling the operator to determine easily the end reaction. The titration should be done rapidly while the solution is hot. If the value of the normal permanganate solution is known in terms of iron, the resulting number of cubic centimeters should be multiplied by the factor .2951, which will give the percentage of manganese. The advantages of this method are that the entire operation is carried on in one vessel, and that ordinary permanganate solution for iron is employed.

When ½ g. of ore is used in the iron determination, a half-normal potassium-permanganate solution is usually employed, and if this same solution is used for the determination of manganese, it will be necessary to take one-half of the number of cubic centimeters of permanganate solution used and multiply it by the factor given. If the solution is neither exactly a normal nor a half-normal it will be necessary to multiply by a factor that will render it normal before employing the manganese factor as already given.

Some operators prefer to remove the heavy precipitate of iron hydrate and the zinc oxide so that the titration can be done in a clear solution. This may be accomplished by taking $1\frac{1}{2}$ g. of the ore and treating it as described, being careful to remove the solution from the hot plate and transfer it to a graduate before adding the zinc oxide. After the zinc oxide has been added, the solution should be made up to 300 c. c. The solution should now be filtered through a dry filter, 200 c. c. of the filtrate being taken, which will correspond to 1 g. of ore. The rest of the filtrate containing the zinc oxide and the precipitated hydrate of iron may be thrown away. The filtrate is then brought to a boil and the titration carried on as before. The fact that the precipitate has a slightly greater specific gravity than the solution will have practically no effect on the accuracy of the results. As this operation requires the treating of larger amounts of ore and the use of several vessels for the determination, it is not so rapid as the one first described.

DETERMINATION OF PHOSPHORUS

16. Emmerton's Method.—The method for the volumetric determination of phosphorus known as Emmerton's is rapid and accurate, provided the necessary precautions are observed. Five g. of ore placed in a casserole is dissolved with 75 c. c. of nitric acid of 1.20 specific gravity. Over the dish a triangle is placed and over this a watch glass. The solution is then boiled to dryness on a sand bath, after which it is heated on an iron plate for 30 minutes, when all free acid should have been expelled. The dish is now removed, cooled, 40 c. c. concentrated hydrochloric acid added, and a watch glass placed on the casserole. The whole is now heated gently and then boiled until about 15 c. c. of the acid is driven off. The dish is cooled and the watch glass washed off with 40 c. c. of concentrated nitric acid, the acid being allowed to run down into the casserole. The dish is then covered with a funnel, and the solution boiled down to 15 c. c. in bulk. The casserole is removed

and its contents moved so as to moisten whatever crust may have formed. The solution is now practically free from hydrochloric acid, and should be diluted with water and washed into a 400-c. c. flask, bringing the bulk up to about 75 c. c. Strong ammonia is added to the solution, the flask being shaken thoroughly after each addition, until a stiff jelly is formed, and a strong smell of ammonia is emitted. Concentrated nitric acid is now added, shaking well after each addition, until the solution thins and shows a dark color, then sufficient nitric acid should be added to bring the solution to a clear amber color. The solution should now have a bulk of about 250 c. c. A thermometer is now inserted in the solution, which is heated or cooled carefully until it has a temperature of 85° C., at which temperature 40 c. c. of ammonium-molybdate solution should be added. The flask should now be closed with a stopper, and wrapped in a thick, warm towel to keep the temperature from varying, and shaken violently for 5 minutes. The yellow precipitate should be collected on a filter, and the flask and precipitate washed with a solution made up of 25 g. of ammonium-sulphate crystals, 50 c. c. of concentrated sulphuric acid, and 2,500 c. c. of distilled water. The washed precipitate is dissolved with ammonia. About 10 g. of granulated zinc is placed in a 500-c. c. flask, the funnel containing the yellow precipitate is inserted in the neck of the flask, and the precipitate washed into the flask with dilute ammonia (1 in 4), using about 30 c. c. only. After the precipitate has been washed with ammonia and twice with water, it should be sucked dry with a filter pump. The ammonia solution is now poured into a small beaker, the funnel is reinserted in the flask, and the solution poured into the beaker through the filter again, the paper being washed thoroughly with water after the ammonia has all run through, finally being sucked dry with the filter pump. Into the flask about 80 c. c. of warm dilute sulphuric acid (1 in 4) is now poured and heated until rapid solution of the zinc commences, and then gently stirred for about 12 minutes, at the end of which time the reduction of the molybdic acid is complete.

This should now be filtered from the undissolved zinc, and the flask rinsed out with cold water poured on the filter. The filtrate should be kept from the air until ready for titration. The solution, which is a dark olive-green, is oxidized with a permanganate solution until colorless, when an extra drop will produce a pink tinge.

17. Calculation of Phosphorus.—In the yellow precipitate of ammonium phosphomolybdate, $(NH_4)_3PO_4 \cdot 12MoO_3 \cdot 3H_2O$, the ratio is $12MoO_3$ to $1P$, or the molecular weight ratio is, in round numbers, 1,728 to 31. The reaction produced by the zinc liberates hydrogen, forming molybdic oxide, as follows:

$$2MoO_3 + 3H_2 = Mo_2O_3 + 3H_2O$$

The reaction with the permanganate solution is:

$$5Mo_2O_3 + 3K_2Mn_2O_8 + 9H_2SO_4 = 10MoO_3 + 3K_2SO_4 + 6MnSO_4 + 9H_2O$$

According to this equation:

$3K_2Mn_2O_8$ corresponds to $10MoO_3$, or $142.94 \times 10 = 1{,}429.4$

$3K_2Mn_2O_8$ corresponds to $30Fe$, or $55.5 \times 30 = 1{,}665.0$

Hence, Fe standard : MoO_3 standard $= 30Fe : 10MoO_3$, or $1{,}665 \div 1{,}429.4 = 1.164$.

The percentage of phosphorus in the yellow precipitate is $\frac{31 \times 100}{1{,}728} = 1.794$ per cent.; hence, the phosphorus $1.794 \div 1.164 = 1.541$ per cent. of the Fe standard.

NOTE.—In making phosphorus determinations by the volumetric molybdate method, a reductor should be used.

The ratio of molybdic acid to iron is $1{,}429.4 \div 1{,}665 = .85849$. The ratio of phosphorus to molybdic acid is $\frac{31}{1{,}728} = .01794$. The value of 1 c. c. of the permanganate solution in terms of iron multiplied by the ratio of molybdic acid to iron and the product by the ratio of phosphorus to molybdic acid gives the value of 1 c. c. of the permanganate solution in terms of phosphorus.

EXAMPLE 1.—A charge of .1498 g. of piano wire (99.27 per cent. metallic iron) requires 42.99 c. c. of permanganate solution to give the

required color. (*a*) What is the strength of the permanganate solution in terms of iron? (*b*) What is the strength of the permanganate solution in terms of phosphorus?

SOLUTION.—(*a*) $.1498 \times .9927 \div 42.99 = .003466$ g. of *Fe* per cubic centimeter of permanganate solution.
(*b*) $.003466 \times .85849 \times .01794 = .0000533$ g. of phosphorus per cubic centimeter of permanganate solution. Ans.

EXAMPLE 2.—The precipitated ammonium phosphomolybdate from 2 g. of ore required 35.5 c. c. of permanganate solution of the strength in example 1. What per cent. of phosphorus does the ore contain?

SOLUTION.— $35.5 \times .0000533 \times 100 \div 2 = .0946$ per cent. of phosphorus. Ans.

18. Handy's Method for Phosphorus.—From 2 to 10 g. of the ore may be employed for this determination, and the process is as follows when only 2 g. is taken: The ore is placed in a beaker with 20 c. c. of HCl and put on a hot plate or sand bath, and kept almost at a boiling temperature until the mixture is reduced to a syrup; then, 20 c. c. of concentrated HNO_3 is added and boiled to about 10 c. c., or until all the HCl fumes are driven off. The sides of the beaker are then washed down and the solution diluted to about 30 c. c. This is then filtered through a rapid filter to get rid of the silica and the filter is washed with as little water as possible. If there is much of the insoluble residue, it may be necessary to fuse it with sodium carbonate and dissolve and add to the filtrate.

The solution is now heated or cooled to 85° C. and 75 c. c. of molybdate solution added. This is shaken for 10 minutes, and then the yellow precipitate filtered off on to a 9 c. m. No. 1 F. filter and washed with a potassium-nitrate solution made by dissolving 1 g. of KNO_3 in 100 c. c. of water. Care must be taken not to allow the phosphomolybdate to crawl over the edges of the filter, and the filter should be filled with the wash to its edge at least six times, so as to be sure to remove all the iron or acid. The filter with the yellow precipitate should now be transferred to a No. 1 beaker and 20 c. c. of sodium-hydrate, $NaOH$, solution added. The filter is opened with a glass rod, the precipitate dissolved, and the

filter paper beaten to a pulp. The solution is diluted with warm water to about 50 c. c. and .5 c. c. of a 1-per-cent. phenol-phthalein solution is added. The solution is now titrated with either HCl or HNO_3 solution, adding it drop by drop near the end of the reaction. The end point is indicated by the solution becoming colorless, and the difference between the number of cubic centimeters of $NaOH$ and the number of cubic centimeters of acid used gives the phosphorus direct. In case the insoluble residue has not been fused, this method does not take into account the phosphorus contained in the silicious matter.

19. Sodium-Hydrate Solution, $NaOH + H_2O$.—To make the sodium-hydrate solution, 15.4 g. of sodium hydrate is dissolved in 100 c. c. of water, and a saturated solution of barium hydrate added, which is stirred in until there is no more precipitate. The barium hydrate is added to free the sodium hydrate from sodium carbonate, which is converted to barium carbonate. The solution should then be filtered and made up to 2 liters by the addition of water.

20. Standard Acid Solution.—The standard acid solution may be either HNO_3 or HCl. In case the nitric acid is employed, 200 c. c. of HNO_3 (sp. gr. 1.42) is diluted up to 2 liters. 200 c. c. of this stock solution can be diluted to 2 liters, and this will form an approximate standard. The standard acid solution may be run against the alkali, and so their relative strengths ascertained. The stronger can be reduced by the addition of water, and in this manner the two can be brought to an equal value.

After the two solutions have been brought to an equal strength, 1 g. of pure ammonium phosphomolybdate may be dissolved in 20 c. c. of alkali solution. This amount of yellow precipitate contains .0016 g. of phosphorus, and if the sodium-hydrate solution were normal, it should require 16.3 c. c. to neutralize the MoO_3 in the yellow precipitate; hence, 3.7 c. c. of acid would be required to neutralize the excess of alkali and change the color of the indicator. In case more or less of the acid is required, the solution can be

brought to the right strength by adding water or by adding more of the alkali until the proper strength is obtained. The acid solution can then be brought to the same strength that the alkali has, and the solution will be correct for working with 2 g. of ore; that is, 1 c. c. will be equal to .0002 g. of phosphorus or $\frac{1}{100}$ per cent. of phosphorus when 2 g. of ore is taken for analysis.

21. Phenol-Phthalein Indicator.—Five-tenths of a gram of phenol-phthalein is dissolved in 200 c. c. of 95 per cent. alcohol; from 3 drops to half a cubic centimeter will be sufficient for each titration.

DETERMINATION OF LIME

22. Treatment of Limestone.—Limestone and ores containing lime are treated in the same manner. 1 g. of ore is treated in a small casserole or beaker with 20 c. c. of distilled water and 5 c. c. of concentrated HCl. This is boiled until the soluble portion of the ore is all in solution and is then diluted and the insoluble residue filtered off, washing it thoroughly with distilled water and adding the washings to the filtrate. If the ore contains any lead, it should be removed by passing sulphureted-hydrogen gas, H_2S, through the solution; the lead will precipitate as lead sulphide, which should be filtered off, and the precipitate well washed. The filtrate should then be heated to boiling and the excess of H_2S oxidized by a few drops of bromine water or a little potassium chlorate. The oxidizer should be added, a little at a time, until the solution is perfectly clear.

Ammonia in slight excess is then added to the filtrate from the sulphide, or, if the ore is free from lead, to the filtrate from the insoluble matter. (The excess may be recognized by the smell or by testing with red litmus paper, which should turn blue if the ammonia is in excess.) The ammonia will precipitate iron and aluminum as hydrates. The solution is then boiled until all the excess of ammonia is expelled. This is necessary, since aluminum hydrate is slightly soluble in an excess of ammonia. To test the

solution to see if all the ammonia is expelled, the tip of a glass rod may be moistened with HCl and held over the solution. If there is any ammonia left in the solution, it will form white fumes with the HCl.

The iron and aluminum hydrates should now be filtered off. This is a very tedious job, as the precipitate is flocculent and jelly-like; it may be shortened by filtering hot, using funnels with long stems—the longer the stem, the greater the suction—and running two or three funnels at once. The precipitate should be washed well with hot water and the washings added to the filtrate. If much iron and alumina are present, the precipitate should be redissolved in a little HCl, reprecipitated with ammonia, filtered, and the filtrate added to the first filtrate. This is advisable, as a very dense iron-aluminum precipitate is apt to carry down a little calcium hydrate with it.

To the filtrate from the hydrates, 1 c. c. of ammonia should be added and the solution brought to a boil. If any iron or aluminum hydrate forms, it should be filtered off. If white magnesium hydrate forms, it should be redissolved by adding a slight excess of HCl, and the solution made again slightly alkaline with ammonia.

The lime is then precipitated as calcium oxalate by adding a solution of ammonium oxalate or oxalic acid. (If oxalic acid is used, the least possible excess should be added, and the solution should contain a considerable excess of ammonia, so that the solution will be alkaline after adding the oxalic acid.) If the ore contains magnesium, a considerable excess of ammonium oxalate should be added, in order to get all the magnesium into the form of magnesium oxalate, which is soluble. This should be again heated nearly to boiling for a few minutes and the calcium oxalate then filtered off. The precipitate should be washed with boiling water until the last trace of oxalic acid is removed. This can be tested by means of a very dilute solution of potassium permanganate, pink or pale purple in color, and made acid with H_2SO_4; the least trace of oxalic acid in the washings will decolorize the permanganate solution.

If the ore contains much magnesium and very accurate work is desired, the calcium oxalate precipitate should be dissolved in HCl and the lime reprecipitated as oxalate and again filtered; otherwise, a little magnesium oxalate is apt to be retained in the precipitate and will cause a high result. The error from this source, however, is usually so small that for ordinary work it may be disregarded.

The filter and its contents may now be removed from the funnel and the precipitate washed off the paper into a beaker with a jet of hot water from a wash bottle. After all the precipitate that can be removed in this way is gotten off the filter, the filter should be washed with dilute H_2SO_4 and the washings added to the contents of the beaker. (Sometimes, it is difficult to remove the last trace of calcium oxalate from the filter by dilute H_2SO_4; in such a case, a few drops of HCl may be added to the paper.) The contents of the beaker should be diluted up to about 100 c. c., and 15 c. c. of sulphuric acid added, the solution heated up to 70° C. (158° F.), and the whole titrated with potassium permanganate.

23. Calculation of Lime.—The lime titration is carried on in the same way as the iron titration with permanganate solutions. The standard of the solution for lime is just half of the standard for iron. The atomic weight of iron is 55.5 and the molecular weight of CaO is 55.68; and as there are at least 2 atoms in a molecule, $\frac{55.68}{2} = 27.84$, or the atomic weight of lime. If 1 c. c. of the permanganate solution equals .01 g. of iron, then it will equal .005 g. of lime. The result of the titration gives the contents of the ore in lime, CaO. Lime contains 71.43 per cent. of calcium and 28.57 per cent. of oxygen, so that the amount of calcium in an ore may be determined, if desired, by multiplying the percentage of lime by .7143.

EXAMPLE.—A solution of potassium permanganate having an iron standard of .012 was used to determine the lime in a limestone. It required 30.2 c. c. of the standard. (*a*) What per cent. of lime was there in the sample? (*b*) What per cent. of calcium? (*c*) What per cent. of calcium carbonate?

SOLUTION.—(*a*) $\frac{30.2}{2} \times .012 \times 100 = 18.12$ per cent. CaO. Ans.

(*b*) $18.12 \times .7143 = 12.94$ per cent. Ca. Ans.

(*c*) $\frac{CaCO_3}{CaO} = \frac{99.25}{55.58} = 1.785$; $18.12 \times 1.785 = 32.34$ per cent. $CaCO_3$. Ans.

DETERMINATION OF COPPER

24. Titration of Copper Ores.—There are three methods in general use for the volumetric determination of copper: the cyanide method, the iodide method, and the sodium-thiosulphate method. The latter method has been adopted by the Western Association of Technical Chemists and Metallurgists.

25. The Cyanide Method.—The cyanide solution should be approximately half normal or 1 c. c. = 5 mg. of copper. Duplicate charges of ore, of 1 g. each (or ½ g. if the ore is very rich in copper), are placed in Erlenmeyer flasks or casseroles, with 7 c. c. of concentrated HNO_3 and 5 c. c. of concentrated H_2SO_4 (commercial acids will answer all purposes). These are boiled until the nitric acid is all expelled and the sulphuric acid is boiling freely and giving off dense, white, sulphurous fumes. Any sulphur in the ore will be partly, or sometimes wholly, volatilized, a part of it recondensing in the neck of the flask. Any sulphur remaining in the bottom of the flask should by this treatment be fused into globules that are yellow when cold and are free from copper. The solution is removed, cooled, and diluted very carefully with water to about 50 c. c., and then 5 g. or 6 g. of commercial sheet zinc cut into strips weighing 2 g. or 3 g. each are added. The flask is now shaken in order to break up any cake that may have formed in the bottom, and is then set aside and allowed to stand for about 10 minutes. By that time all the copper will have been precipitated. 50 c. c. of water and 20 c. c. of concentrated H_2SO_4 are then added, to dissolve rapidly the excess of zinc. As soon as the zinc is all dissolved, the flask is filled up to the neck with water ("tap" water will answer for this purpose); the copper is allowed to settle out, and the water is then poured off

very carefully, leaving the copper behind. This operation is repeated twice more, to thoroughly wash out the zinc sulphate. The last water is now poured off, 5 c. c. of concentrated HNO_3 is added (chemically pure acid being used), and the copper dissolved in the same way as the foil for standardizing was dissolved. From here on the operations are exactly the same as in the standardization of cyanide solutions. A 50-c. c. burette is filled with potassium-cyanide solution, and the cyanide solution dropped gradually into the copper solution until the blue color disappears and the solution becomes colorless. The number of cubic centimeters of potassium-cyanide solution required to do this is read from the burette and then the quantity of copper in the ore is calculated. It is usually advisable to filter the copper-hydrate solution, either before titrating or when all but about 2 or 3 per cent. of the copper has been neutralized. The object of this filtration is to remove the gangue, lead, ferric hydrate, etc. that may be present, and afford a clear solution with which to complete the titration.

If the ore contains silver, a *drop* of HCl should be added to the HNO_3 solution before adding the ammonia; if there is very much silver, 2 drops may be necessary; any unnecessary excess should be avoided. The precipitated chloride of silver is then filtered off, the ammonia added, and the solution titrated as usual.

Strips of aluminum may be used instead of zinc for precipitating the copper. The precipitated copper may be dissolved off the aluminum with nitric acid and the strips washed and used over and over again until they get too thin to handle.

If an ore refuses to go into solution with HNO_3 and H_2SO_4, a few drops of HCl added will usually suffice to decompose it. Both the HCl and the HNO_3 must be thoroughly expelled before precipitating the copper, otherwise they will hold a little copper in solution and cause a low result.

26. The Iodide Method.—By the iodide method, 1 to 3 g. of ore is treated, in a small flask, with 20 c. c. of

concentrated nitric acid, 5 c. c. of concentrated hydrochloric acid, and 5 c. c. of concentrated sulphuric acid. The solution is then heated gently and evaporated to sulphur fumes, after which it is cooled and 20 c. c. of water added, the whole being heated to boiling or until all copper is in solution. The $SiO_2 \cdot PbSO_4$ is filtered out and the filtrate allowed to run into a small beaker. The casserole is now washed and filtered with hot water, the volume of the filtrate and washings being kept down to about 50 c. c. Two pieces of sheet aluminum $\frac{1}{16}$ inch thick and $1\frac{1}{2}$ inches square are then placed in the beaker, and 5 c. c. of strong sulphuric acid added; the beaker is now covered and boiled 10 minutes, or until all copper is precipitated. The solution and copper are poured into a clean beaker, allowed to settle, and then decanted through a filter and washed with hot water; the washings are then poured through a filter, as much as possible of the copper being left in the beaker. To the beaker containing the aluminum to which some copper may adhere, 6 c. c. of nitric acid of 1.2 sp. gr. is added and the whole warmed until the copper has dissolved. The beaker containing the copper is now placed under the funnel and this last solution is poured through the filter. The beaker is now heated until all the copper has dissolved, then $\frac{1}{2}$ g. of potassium chlorate is added and the solution boiled a few minutes to oxidize any arsenic precipitated with the copper. The beaker is now replaced under the funnel and the filter and beaker containing the aluminum washed into it with hot water. The copper is now in the beaker as nitrate, and to this is added 20 c. c. of zinc acetate. The solution is finally titrated as in standardizing iodide solutions and the copper calculated in the same manner. See *Assaying*, Part 4.

27. The Sodium-Thiosulphate Method.—In the sodium-thiosulphate method, .5 g. of ore is dissolved in 2 c. c. of nitric acid, 3 c. c. of hydrochloric acid, and 4 c. c. of sulphuric acid. This solution is evaporated to copious white fumes, then cooled and diluted with 25 c. c. of cold water. A piece of sheet aluminum is added and the solution boiled

until all the copper has been precipitated. To insure complete precipitation of the copper, 10 c. c. of hydrogen-sulphide water is added. The copper is not generally precipitated completely by aluminum—hence the need of hydrogen-sulphide water. Arsenic and antimony are precipitated by the hydrogen-sulphide water, and are oxidized by bromine; otherwise, the results would be high. The solution is now filtered, and washed three times by decantation, when 55 c. c. of strong nitric acid is poured over the aluminum and copper. The aluminum is removed and washed with a minimum of hot water. The beaker is placed under the funnel and strong bromine water poured through the filter to dissolve any copper or sulphide of copper that may have been left over. The filter is now washed with hot water, and then the solution in the beaker evaporated to 2 or 3 c. c.; 5 c. c. of hot water, and 6 c. c. of strong ammonia added and the whole boiled. To this, 8 c. c. of acetic acid and 10 c. c. of the potassium-iodide solution are added, and the whole shaken until all of the copper is precipitated. The solution is finally titrated, the same as in standardizing thiosulphate solutions. See *Assaying*, Part 4.

DETERMINATION OF LEAD IN ORES

28. Titration of Lead Ores.—Lead may be determined accurately and comparatively quickly by titration with a standard solution of potassium ferrocyanide, or by the ammonium-molybdate method. The ferrocyanide solution should be such that 1 c. c. will precipitate 10 mg. of lead as ferrocyanide of lead. The solution is standardized by treating solutions containing a known quantity of lead. It requires three solutions for titrating by the ammonium-molybdate method as follows:

1. A solution of ammonium molybdate containing 8.64 g. of the molybdate to the liter.

2. A solution of ammonium acetate containing 200 g. to the liter.

3. A solution of tannic acid containing 1 part of the acid in 300 parts of water. This should be made fresh every few days.

29. Ferrocyanide-of-Potassium Titration for Lead. In the titration for lead with ferrocyanide of potassium, duplicate 1-g. charges of ore (or ½-g. charges if the ore is rich in lead) are treated in casseroles, with 15 c. c. of concentrated HNO_3 and 10 c. c. of concentrated H_2SO_4. This is boiled until the nitric acid has been completely expelled and the sulphuric acid gives off dense, white, sulphurous fumes, after which it is removed and cooled. The solution is then diluted very carefully with cold distilled water; stirred thoroughly to break up clots; and then boiled until all soluble sulphates are in solution. The mixture is now filtered, as much as possible of the residue being left in the casserole, and then this residue is washed twice with hot dilute sulphuric acid (very dilute—1 per cent. concentrated acid) and once with cold water, the washings being poured on the filter. The precipitate on the filter is now washed back into the casserole, as little water as possible being used, and then 30 c. c. of a saturated solution of ammonium carbonate is added. (A solution of any salt is saturated when it will dissolve no more of the salt.) This is now heated quickly to boiling, with only a screen between the flame and the casserole, and boiled for a minute, in order to decompose any calcium sulphate that may have formed along with the lead sulphate; otherwise, the calcium will react on the lead and cause a low result. The solution is filtered and the precipitate washed thoroughly with hot water containing a little ammonium carbonate. The carbonate of lead on the filter paper is dissolved with strong chemically pure acetic acid, diluted to 180 c. c., and titrated. The results are figured as usual in volumetric work. The average result of the two titrations is taken.

30. Ammonium-Molybdate Titration for Lead.—In the titration for lead with ammonium molybdate, .5 g. or 1 g. of ore is dissolved in 10 c. c. of strong nitric and 10 c. c. of strong sulphuric acid. This is then evaporated to white fumes, cooled, diluted to 50 c. c., and boiled, after which it is decanted on the filter paper and washed by decantation three or four times with hot water containing 2 per cent. of

sulphuric acid, and then once with a little cold water. Now, with the aid of the wash bottle, the sulphates are transferred from the filter back to the beaker, 50 c. c. of ammonium-acetate solution is added, and the whole boiled thoroughly. This procedure is necessary on account of the fact that lead sulphate is not readily soluble in ammonium acetate when other sulphates such as barium and calcium are present. The solution is now filtered again through the original filter, washed with hot water (the filtrate being made slightly acid with acetic acid) and titrated, as in standardizing molybdate solutions. See *Assaying*, Part 4.

DETERMINATION OF ZINC

31. Titration of Zinc Ores.—Zinc is commonly determined volumetrically by means of a standard solution of potassium ferrocyanide, which precipitates zinc as ferrocyanide of zinc. Duplicate 1-g. charges of ore are treated in casseroles with 15 c. c. of aqua regia (2 parts of HCl and 1 part of HNO_3), evaporating nearly to dryness. When the contents of the casserole are nearly dry, 25 c. c. is added of a solution of potassium chlorate in nitric acid, made by shaking an excess of crystals of chemically pure potassium chlorate with concentrated HNO_3 in a flask. (This solution must be handled with care and kept in an open flask, as it is a violent explosive.) The solution is added very gradually and the casserole warmed gently, without covering, until violent action ceases and no more greenish fumes are given off. It is then covered with a watch glass and run down to dryness. As soon as the mass is dry, and the last drop of acid condensed on the watch glass has evaporated, the casserole is removed from the hot plate, as overheating or baking the residue will cause more or less error. The casserole is then cooled and 7 g. of chemically pure ammonia chloride, 15 c. c. of strong ammonia, and 25 c. c. of hot distilled water added. This is now boiled for 1 minute, being stirred meanwhile with a rubber-tipped glass rod, to break up any clots; and any splashes rubbed off that may have dried on the sides

and cover. The solution is then filtered into a 250-c. c. flask and the residue washed several times with a hot solution of ammonium chloride, made by dissolving 10 g. of chemically pure ammonium chloride in 1 liter of distilled water and adding a few drops of ammonia to make the solution alkaline. Any iron and aluminum in the solution will be precipitated as hydrates by the ammonia. If the precipitate is heavy, it is liable to carry down some zinc hydrate with it, and should be filtered out, washed well, transferred to a casserole (using as little water as possible), evaporated down to dryness, and then treated with the solution of potassium chlorate in nitric acid in the same manner as the ore was first treated. This is run down to dryness, and again treated with ammonium chloride and ammonia, as in the case of the ore; the precipitated iron and aluminum hydrates are again filtered off, and washed with the hot ammonium-chloride solution, the filtrate and washings being added to the first filtrate. If the first precipitate of hydrates is small, however, the little zinc it carries down may be neglected. The precipitate is filtered out, washed as before described, and the operation continued. If the filtrate is blue, it contains copper, which *must* be removed before titrating. Enough HCl is added until the solution is neutral (indicated by the disappearance of the blue color if copper is present, or by blue litmus paper turning red if the solution contains no copper) and then 10 c. c. excess of concentrated HCl is added. If the solution contains copper, 20 to 40 g. of test lead is added and the solution boiled until all the copper is precipitated. The solution is decanted from the lead into a beaker, the lead and precipitated copper is washed thoroughly (the washings being added to the main solution) and the whole then titrated as in the standardization.

If, as may sometimes happen, the ore will not decompose completely by treatment with aqua regia, the solution should be evaporated to dryness, the soluble salts dissolved out with water, and the insoluble residue filtered off. The insoluble residue is fused in a porcelain crucible with carbonate and nitrate of soda, the fused mass is dissolved with water and

nitric acid, the silica filtered off, and the filtrate added to the first filtrate. The whole is now evaporated nearly to dryness and the operation proceeded with as before.

32. Western Chemical Society's Zinc Method.—The method of determining zinc adopted by the Western Chemical Society is here given. The solutions employed are as follows:

A solution of ammonium chloride, made by dissolving 60 g. of the salt in 250 c. c. of water and adding 150 c. c. of strong ammonia.

A solution of potassium ferrocyanide, containing 21.6 g. of the pure salt to the liter of water.

A solution of uranium acetate, 4.5 g. to 100 c. c. of water and 2 c. c. of acetic acid.

A saturated solution of potassium chlorate in nitric acid.

The method of standardization is as follows:

.2 to .3 g. of freshly ignited zinc oxide is dissolved in 10 c. c. of hydrochloric acid, and is made just neutral with ammonia. To this, 6 c. c. of concentrated hydrochloric acid and 180 c. c. water is added, the whole being then boiled and titrated. The results are checked, chemically pure, zinc being used instead of zinc oxide for the purpose.

33. The solution is now divided into two equal parts, and one part quickly titrated, the remainder being then added and enough run in quickly to finish the titration almost; 2 drops at a time are then added, the solution being tested after each addition by placing a drop of the solution on a drop of the uranium acetate on a spot plate. When this gives a yellow color, the titration is finished. By taking care to complete the titration by adding only 2 drops of ferrocyanide at a time, a delicate end reaction may be obtained. If several successive spots develop the yellow color, it is then possible to deduct from the burette reading the value of the number of spots added after the end color was first obtained.

34. Treatment of Zinc Ores.—In the treatment of zinc ores, from .5 g. to 1 g., according to the richness of the ore, is dissolved in 4 to 8 c. c. of hydrochloric and an equal

amount of nitric acid, which is then evaporated to one-third of the original volume. If gelatinous silica is present, the solution should be diluted a little and filtered. (Gelatinous silica combines with zinc in an alkaline solution.) To this, 15 c. c. of the solution of potassium chlorate in nitric acid is added.

The solution is now evaporated to dryness, care being taken not to bake it (if the dry mass is heated much above the boiling point, the zinc is rendered difficultly soluble in ammonium chloride; furthermore, chloride is volatile, in the presence of hydrochloric acid, at a comparatively low temperature). The mass is cooled, 40 c. c. of the ammonium-chloride solution added and the solution boiled, care being taken that all clots are broken up. This is filtered and washed with a hot solution of ammonium chloride. If much iron is present, the precipitate should be dissolved in a minimum of hot dilute ($\frac{1}{3}$) hydrochloric acid, and the treatment with ammonium-chloride solution repeated. (It is very difficult, if not impossible, to wash all the zinc out of the ferric hydroxide.)

The filtrate is neutralized with 15 c. c. of strong hydrochloric acid and 15 g. of granulated lead added, and the solution boiled until all copper is precipitated, when titration is performed.

The amount of solution, of free hydrochloric acid, and of ammonium chloride should always be the same, and the solution should always be titrated at the same temperature, which should be near the boiling point.

DETERMINATION OF SULPHUR

35. Elliott's Method.—The volumetric determination of sulphur depends on:

1. The treatment of the material containing sulphur with an acid that will liberate the sulphur.
2. The formation of hydrogen sulphide, H_2S, and its absorption in a solution of sodium hydrate.
3. The liberation of sulphur from hydrogen sulphide by titration with iodine.

In order to carry out this method for rapidly determining sulphur by volumetric analysis, the following solutions are required:

1. A bichromate-of-potassium solution of 5 g. of pure $K_2Cr_2O_7$ in 1,000 c. c. of water. Theoretically, such a solution is equivalent to .00569 g. of iron; it should, however, be titrated to ascertain its equivalent in iron.

2. A thiosulphate solution containing 24.8 g. of $Na_2S_2O_3 \cdot 5H_2O$ in 1,000 c. c. of water. Theoretically, such a solution is equivalent to .01265 g. of iodine; it should, however, be standardized.

3. An iodine solution containing 12.65 g. of potassium iodide in 1,000 c. c. of water. Theoretically, such a solution is equivalent to .0248 g. of thiosulphate solution.

4. A solution containing 1 g. of starch to 150 c. c. of boiling water.

36. Standardizing the Solutions.—The method of standardizing the bichromate of potassium has been explained, and will not be repeated; however, the reaction that occurs is shown by the equation:

$$6FeCl_2 + K_2Cr_2O_7 + 14HCl$$
$$= 3Fe_2Cl_6 + 2KCl + Cr_2Cl_6 + 7H_2O$$

From this equation, one equivalent of $K_2Cr_2O_7$ with a molecular weight of 292.26 is equal to six equivalents of iron with a molecular weight of 333.

When bichromate of potassium is added to a solution of iodide of potassium in the presence of free HCl, iodine is liberated according to the equation:

$$K_2Cr_2O_7 + 6KI + 14HCl = 8KCl + Cr_2Cl_6 + 7H_2O + 6I$$

From this equation, it is seen that one equivalent of $K_2Cr_2O_7$ with a molecular weight of 292.26 liberates six equivalents of iodine with an atomic weight of 756.06.

It is evident from this that, by adding to a solution of iodide of potassium in the presence of HCl, a known quantity of bichromate solution, the absolute amount of iodine liberated can be calculated. When a solution of iodine is added to a solution of thiosulphate containing a little starch,

a permanent blue color will appear when an excess of iodine occurs; if, therefore, a solution containing a known amount of iodine is titrated with a thiosulphate solution, the latter may be standardized, the reaction being as follows:

$$2Na_2S_2O_3 \cdot 5H_2O + 2I = 2NaI \cdot 4H_2O + Na_2S_4O_6 + H_2O$$

If the iodine is used to titrate a solution containing free hydrogen sulphide, H_2S, also containing starch, the reaction would be complete when a blue color had been formed, and if the amount of iodine is known the amount of sulphur liberated can be calculated. The reaction may be expressed as follows:

$$H_2S + 2I = 2HI + S$$

37. Calculating Elliott's Method.—Suppose that by titration it was found that 1 c. c. of bichromate solution was equivalent to .0566 g. of iron; then,

$$333 : 292.26 = .00566 : .004967 \text{ g. of } K_2Cr_2O_7$$

But, $$292.26 : 756.06 = .004967 : .001284$$

that is, 1 c.c. of the bichromate solution will liberate .001284 g. of iodine. Suppose, on titration, it was found that 1 c. c. of thiosulphate will decolorize .01267 g. of the iodine; then, 10 c. c. of the thiosulphate solution is measured out and diluted to 300 c. c., when a few drops of starch solution are added, and the whole titrated. Suppose it required 20.1 c. c. of the iodide solution to give a permanent blue color; then, 20.1 c. c. = .1267 g. of iodine, or 1 c. c. $= \frac{.1267 \times 10}{20.1}$ $= .006303$ g. As it requires 2 parts of iodine to liberate 1 part of sulphur according to the formula H_2S, the proportion is, for sulphur, $2I : S = 252.02 : 31.82 = .006303 : .000795$, or 1 c.c. of iodine is equal to .000795 g. of sulphur.

GRAVIMETRIC ANALYSIS

38. Gravimetric Assaying.—In wet assaying, the substance under examination is taken into solution by a liquid chemical and then precipitated. The reagents added will precipitate the element in the form of an insoluble compound of known composition. This precipitate is then

separated from the solution by filtering, and is washed, dried, and weighed. The composition and weight of the precipitate being known, the weight of the element in question may be calculated. This method is known as **gravimetric analysis,** or analysis by weighing. It is frequently necessary in gravimetric analysis to fuse one substance with another in order to make a soluble compound, from which an insoluble precipitate of the substance sought can be obtained. Gravimetric analysis, while slower than volumetric, is practiced where an exhaustive analysis of a mineral is required, and in cases where volumetric analysis is not reliable.

39. Determination of Iron in Iron Wire.—The determination for iron in iron wire, and that for iron and steel, can be more quickly and as accurately accomplished by the volumetric method. Two samples of bright iron wire of approximately .5 g. are weighed, transferred to No. 3 beakers, and digested with 25 c. c. of hydrochloric acid and 5 c. c. of nitric acid. The beaker is covered and gently warmed until solution is complete. The samples are both treated alike. Next, 100 c. c. of water is added and ammonia then gradually added until the solution is slightly alkaline; this is boiled and filtered through a No. 4 ashless filter, and the precipitate washed with hot water until the water coming off no longer has an alkaline reaction. The precipitate is dried at 105° C. As much as possible of the dry precipitate is removed to a piece of glazed paper. The filter paper is ignited in a weighed porcelain crucible, uncovered until all carbonaceous material is expelled. The precipitate from the glazed paper is then added, the crucible covered, and the whole ignited to red heat for 10 minutes, after which it is cooled in a desiccator and weighed.

The crucible and contents are heated a second time, cooled, and weighed, and then a third time if the weight is not constant.

EXAMPLE.—Wire taken, .5 g. Weight of crucible, 8.721 g. Weight of crucible and ferric oxide, Fe_2O_3, 9.432 g. Weight of ferric oxide, $9.432 - 8.721 = .711$ g. What was (*a*) the weight, and (*b*) the percentage of iron in the wire?

SOLUTION.—(*a*) $Fe_2O_3 : Fe_2 = .711 : x$, or $158.64 : 111 = .711 : x$; hence, $x = \frac{111 \times .711}{158.64} = .4974$ g. of *Fe* in the wire. Ans.

(*b*) $\frac{.4974 \times 100}{.5} = 99.48$ per cent. of *Fe* in the wire. Ans.

40. Determination of Copper in Copper Sulphate. To determine the amount of copper in copper sulphate, $CuSO_4 \cdot 5H_2O$, about 5 g. of copper sulphate is pulverized, pressed between a filter, and placed in a small weighing tube, which is then sealed. The tube and its contents are weighed, and then about 1 g. of the sulphate is poured from the tube into a beaker, and the tube reweighed. The difference between the two weights of the tube gives the weight of the sulphate in the beaker. Copper sulphate is weighed in this manner, so that it will not injure the scales or scale pan.

The salt is dissolved in 100 c. c. of hot water, and, if the solution is not clear, a few drops of dilute sulphuric acid are added. This is then warmed and a clear solution of sodium hydrate, $NaOH$, added gradually, with constant stirring, until the copper solution is alkaline. It is then boiled, when the copper will be precipitated as a brown cupric oxide; thus:

$$CuSO_4 + 2(NaOH) = CuO + Na_2SO_4 + H_2O$$

The precipitate is allowed to settle, and the precipitate decanted on a No. 4 ashless filter, and washed with hot water until the reaction of the washings is no longer alkaline. This is now dried at 105° C. The precipitate is removed from the filter paper, and placed on a piece of glazed paper. The filter paper is transferred to a weighed porcelain crucible and ignited. This is allowed to cool, when 2 or 3 drops of nitric acid are added to dissolve the copper that was reduced from the cupric oxide by the carbon of the filter paper. When the solution is complete, it is evaporated to dryness and heated to redness, to convert all copper nitrate to cupric oxide. The cupric oxide on the glazed paper is now added to that in the crucible, and the whole heated to red heat several times, or until a constant weight is obtained.

EXAMPLE FOR PRACTICE

Suppose that the weight of the tube and copper sulphate was 7.025 g.; that the second weighing was 5.9605 g.; and that the cupric oxide, CuO, and crucible weighed 15.3744 g. and the crucible alone 15.036 g. (*a*) What would be the weight of the copper sulphate taken? (*b*) What would be the weight of the copper in the sulphate? (*c*) What per cent. of copper was in the sulphate?

Ans. $\begin{cases} (a) \ 1.0645 \text{ g. of } CuSO_4 \\ (b) \ .2702 \text{ g. of } Cu \\ (c) \ 25.38 \text{ per cent. of } Cu \end{cases}$

NOTE.—It is generally the custom to calculate the theoretical quantities and compare them with the results obtained; thus, in the above example,

$$CuSO_4 + 5H_2O : Cu = 100 : x$$
$$247.84 : 63.1 = 100 : x$$

and $$x = \frac{63.1 \times 100}{247.84} = 25.46 \text{ per cent. of } Cu,$$

thus giving a difference of .08 per cent. from that found by analysis.

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41. Determination of Lead in Galena.—To determine lead in galena, 2 g. of finely powdered ore is placed in a 4-inch porcelain crucible and 25 c. c. of warm nitric acid, and 15 c. c. of sulphuric acid are added. This is evaporated carefully until all red nitrous fumes cease. When cooled, a few drops of dilute sulphuric acid and 75 c. c. of water are added, the whole being then brought to a boil, filtered, and washed well. The filtrate may be neglected. The precipitate is washed from the filter into a No. 3 beaker, not over 75 c. c. of water being used; then, 100 c. c. of sodium-carbonate solution (1 to 10) is added and the contents of the beaker boiled for 15 or 20 minutes. The solution must be strongly alkaline, in order to convert the lead sulphate into carbonate. The solution is now filtered and washed well with hot water until the reaction of the wash filtrate is no longer alkaline. The filtrate may be neglected. The precipitate is washed into a No. 3 beaker with about 75 c. c. of water, then 75 c. c. of strong acetic acid is added, and the contents of the beaker kept at boiling temperature for 10 minutes, and covered with a watch glass. Lead carbonate is converted into soluble lead acetate, while silica or gangue remains insoluble. This is now filtered and washed well with hot water, the residue on the filter being

neglected. To the solution of lead in the beaker, which should not exceed 200 c. c., dilute sulphuric acid is added until no precipitate is formed. After standing for half an hour the lead sulphate is filtered off on a No. 3 ashless filter, washed with hot water, and dried at 102° C. The dry lead sulphate is then transferred to glazed paper, and the filter paper ignited in a weighed porcelain crucible. After incineration, a few drops of nitric acid are added and the solution is warmed gently in order to reduce any metallic lead to lead nitrate, and then 3 or 4 drops of sulphuric acid are added and evaporated to dryness. The rest of the lead sulphate on the glazed paper is added, and the contents of the crucible ignited to redness. This is then cooled in a desiccator, and weighed. The igniting and cooling are repeated until the assay has a constant weight.

EXAMPLE FOR PRACTICE

First weighing of tube and galena	16.670 g.
Second weighing of tube and galena	14.503 g.
Crucible + $PbSO_4$ weighed	17.576 g.
Crucible	16.564 g.

(*a*) What was the weight of galena taken?
(*b*) What was the weight of $PbSO_4$ found?
(*c*) What was the weight of the lead found?
(*d*) What percentage of lead was in the galena sample?

Ans. (*a*) 2.167 g.; (*b*) 1.012 g.; (*c*) .6914 g.; (*d*) 31.9 per cent.

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42. Determination of Sulphur.—The following method will be found useful for the determination of sulphur in practically any compound: one gram of the substance is fused with from one to two sticks of potassium hydrate, KOH (the chemically pure caustic potash by alcohol should be used, as any other generally contains sulphur, and even this should be tested to be sure that no sulphur is present) in a platinum or silver crucible (a crucible lined with gold is to be preferred, as the alkali generally attacks the other

crucibles to a slight extent) over a spirit lamp. The best method of making the fusion is to place the *KOH* in the crucible and heat over the spirit lamp until the fused mass is quiet. The lamp is removed from under the crucible, the substance brushed into it, and heated for from 5 to 30 minutes, or until the substance is thoroughly decomposed. (A spirit lamp is used on account of the fact that gas always contains more or less sulphur, and were the gas flame employed, sulphur would be introduced into the material in the crucible.) After the material in the crucible is thoroughly fused, it should be removed from the flame and allowed to cool. As soon as cold, the mass should be dissolved in a beaker with warm water, and when it is all transferred to the beaker it is boiled and filtered. The filter is washed with boiling water until the washings come through free from sulphides or sulphates. To the filtrate from 20 to 40 c. c. of bromine water is now added, the whole heated to about 90° C., and then acidified with *HCl*. If the substance contains silica, it will now be in solution and must be removed by evaporating to dryness, heating the dry material to render the silica insoluble, and then taking up the remainder of the material with water and *HCl*, after which the silica, which has been rendered insoluble, may be filtered off. The filtrate from the silica is boiled and a boiling solution of barium chloride, $BaCl_2$, added until all the sulphur is precipitated as barium sulphate, $BaSO_4$. By heating the barium-chloride solution before adding it to the solution, the barium sulphate will be precipitated almost immediately, while if the solution were cold it would take some time. After the addition of barium chloride, the solution is brought to a boil and then removed to a warm place and allowed to settle. When the precipitate has settled to the bottom, the precipitate is washed and filtered thoroughly with boiling water; then, a few drops of dilute *HCl* are dropped around the edge of the filter paper, which is then washed twice more with hot water. The last washing should be tested with silver nitrate to be sure that all the *HCl* has been removed. The object of the last washing with *HCl* is to insure the removal of all the calcium salts.

The precipitate should now be dried by removing the funnel and filter paper to a ring stand or some suitable support and warming until the precipitate is dry. It may then be removed from the filter paper by placing a platinum crucible on a sheet of glazed paper or a clean watch glass and gently rolling the filter paper between the fingers over it in such a manner as to rub off the precipitate. After all the precipitate that it is possible to remove has been transferred to the crucible, the filter paper should be rolled up, placed on the platinum lid of the crucible, and burned by holding the lid over the flame of a burner or spirit lamp. The ash of the filter paper is then added to the contents of the crucible and the whole ignited in a muffle or over a blast lamp. The crucible is then cooled, when its contents should be perfectly white. The precipitate may be weighed either by transferring from the crucible to the watch glass of a balance or by weighing with the crucible. The weight of the barium sulphate, less the known weight of the ash of the filter paper, multiplied by .13734, will be the weight of the sulphur present in the amount of the substance taken.

When silica is not present, the evaporation of the filtrate from the solution of the fusion can be omitted, which will greatly shorten the method. When evaporating to dryness, care must be taken on account of the fact that, if it is done too rapidly, some of the material is liable to be lost by spurting.

43. Determination of Phosphorus.—In the gravimetric determination of phosphorus, from 2 to 10 g. of the ore to be tested is dissolved in HCl. When only 2 g. is employed, 20 c. c. of HCl will be sufficient. It is best to keep the solution on a sand bath or hot plate almost at the boiling point until it is reduced to a syrup, and then add 20 c. c. of concentrated HNO_3. The solution should be boiled down to about 10 c. c. or until no more fumes of HCl are given off.

In case organic matter is present, it will be necessary to destroy it by carrying the evaporation with HNO_3 to dryness

and then baking the residue until no more acid fumes are evolved. When the material becomes cool, it should be dissolved in HCl and then evaporated down to a syrup as before, HNO_3 added, and the whole evaporated until no more HCl fumes are given off.

The sides of the beaker are now washed down, and the solution diluted to about 30 c. c. This is now filtered through a rapid filter to get rid of the insoluble residue, which, if it is present in considerable quantity, should be ignited with sodium carbonate, the fused mass dissolved in dilute H_2SO_4 and then added to the HNO_3 solution. The solution should contain as little free acid as possible.

To this, 35 c. c. of concentrated NH_4OH is now added, which will precipitate any iron and aluminum present and render the solution alkaline. Then, about 37 c. c. of concentrated HNO_3 is added; this is to dissolve the precipitate and clarify the solution, thus rendering it slightly acid. If it does not, a trifle more of the acid should be added. The reaction between the ammonia and the nitric acid will usually heat the solution to about 85° C. If, on testing with a thermometer, the temperature is not 85° C., the solution should be heated or cooled to this temperature, when it should be taken off the sand bath. While still hot, 75 c. c. of molybdate solution is added. This may be added while the material is in a large beaker or in an Erlenmeyer flask. The thermometer used in ascertaining the temperature of the solution should be washed off by means of the molybdate solution as it is added.

If arsenic is known to be present, the temperature of the solution should be reduced to 25° C. before the molybdate solution is added and the material allowed to stand at least 4 hours with occasional agitation, when the phosphorus will be precipitated and the arsenic will remain in solution.

Where arsenic is not present, the molybdate solution is added while the ore solution is hot, as before stated, and the precipitation is assisted by shaking or agitating the solution. If the solution is in a flask or glass-stoppered bottle, it may be shaken mechanically, or if in a beaker or flask, it may be

given a rotative motion by whirling or rotating the flask in the hand. If the laboratory is provided with compressed air, the shaking may be accomplished by carrying a current of air into the solution through a glass tube and allowing it to ascend to the surface as bubbles.

After the phosphorus has been thoroughly precipitated, it should be filtered off. Care must be exercised in the filtering and in the subsequent washing not to allow the yellow precipitate to crawl over the upper edges of the filter. While the precipitate is on the filter, it should be washed with a 2-per-cent. nitric-acid solution filling at least six times to its edges. This is to remove all iron.

The yellow precipitate can now be dissolved in ammonia, and if any of the precipitate should adhere to the flask or beaker in which the precipitating was accomplished, this may be dissolved in a portion of the ammonia employed for bringing the material on the filter into solution. The solution should now be made acid with dilute sulphuric acid and then alkaline with ammonia in excess. This should be cooled to at least 25° C., and when cold 50 c. c. of magnesia mixture should be added. The flask or beaker should be allowed to remain in a cool place for at least 4 hours, with frequent agitations. By this time all the phosphorus should be precipitated, and it then run on to a filter, the weight of the ash of which is known. The resulting magnesium pyrophosphate and ash are then washed with water, ignited, and weighed. The known weight of the ash of the filter is then subtracted from the weight obtained and the result multiplied by .27928, which will give the weight of phosphorus in the amount of the substance taken, and from this the percentage can be easily figured. It is usually best to test the solution before filtering by adding a few drops of magnesia mixture and allowing the solution to stand for a few minutes in order to see whether all the phosphorus has been precipitated. If a precipitate results from the addition of more magnesia mixture, it is evident that all the phosphorus has not been thrown down, and the solution must be allowed to stand longer.

44. The Magnesia Mixture.—The magnesia mixture may be made by dissolving 1 g. of magnesium sulphate, $MgSO_4$, and 1 g. of ammonium chloride, NH_4Cl, in 4 c. c. of ammonia, NH_4OH, and 8 c. c. of water. One c. c. of this solution will precipitate .024 g. of phosphoric acid, P_2O_5. The magnesium sulphate and the ammonia chloride should be in the form of salt.

45. The Molybdate Solution.—This may be made by dissolving 1 g. of molybdic acid, MoO_3, in 4 c. c. of ammonia and then pouring the solution into 15 c. c. of nitric acid (sp. gr. 1.2). One c. c. of this solution will precipitate .0013 g. of phosphorus.

In making either the magnesia or the molybdate solution, the proportions will be increased so as to make a sufficient volume of the material required. Either one of the solutions should be allowed to stand for a day or two before it is used and then the clear solution siphoned off. In making up the molybdate solution, it will be necessary to keep the flask immersed in cold water, in order to keep the temperature down while adding the ammonia solution to the nitric acid.

DETERMINATION OF SILICA

46. Insoluble Matter.—The insoluble residue left after treating an ore with acids usually consists principally of silica, together with other substances that are insoluble in the acids used. This residue is spoken of as *insoluble matter*.

To determine the proportion of insoluble matter in an ore, 1 g. of the ore may be treated with acid, as in the iron and lime determinations, and as soon as all of the soluble portion of the ore is in solution, the insoluble residue should be diluted with water and filtered off on a special filter paper, the weight of the ash from which is known. The residue should be washed thoroughly on the filter with hot distilled water, and then the filter and its contents placed in a weighed porcelain or platinum crucible and heated over a Bunsen

burner until the moisture has all evaporated, when they are ignited in the muffle or in the flame of a blast lamp, and run down to ash. The crucible and contents should be cooled and then weighed. The gain in weight of the crucible minus the weight of the filter ash is the weight of the insoluble matter.

47. Silica and Its Associates.—The insoluble portion of an ore does not always consist of silica; in fact, it may contain no silica at all. There are various other substances that are insoluble in ordinary acids—for instance, lead and barium, oxides of tin, chromium, titanium, and aluminum—and these when present in the ore make up the insoluble residue.

48. Lead sulphate, if fused with sodium bicarbonate, would be liable to be reduced to metallic lead. Consequently, if any lead sulphate is present in the insoluble residue, it must be removed before performing the fusion. Lead sulphate and all other lead salts are readily soluble in a hot solution of ammonium acetate. The solution is best made by adding strong acetic acid to strong ammonia water until the solution is just acid (indicated by the reddening of a piece of litmus paper placed in the solution) and then adding a few drops of ammonia—just enough to neutralize the excess of acid and make the solution again slightly alkaline (turn the litmus paper blue). This solution should be heated nearly to boiling, and then with the residue should be washed on the filter several times. The hot ammonium acetate will dissolve out the lead sulphate. The residue on the filter is washed several times with hot distilled water, and then transferred to a platinum crucible, and run down to ash as before.

After the filter paper is completely burned, 2 or 3 g. of bicarbonate of soda (some chemists prefer to use a mixture of equal parts of sodium and potassium bicarbonates or carbonates) is added and the whole fused over the blast lamp or in the muffle until all action has ceased and the fusion becomes perfectly liquid and quiet. The fusion will generally

require from 10 to 20 minutes. The heat should be raised as high as possible during the last few minutes.

When the fusion is complete, the crucible should be removed and its bottom dipped into cold water to chill the contents quickly. It is a good plan to place in the crucible before chilling a piece of heavy platinum wire, bent into a hook at the lower end; the fused material will solidify around this hook, and the wire can then be used as a handle with which to remove the fusion from the crucible after it has become loosened. Boiling water from a wash bottle is added to loosen the fusion so that it may be removed. Slightly bending the crucible a few times with the fingers will assist in loosening the fusion, and will not injure the crucible. A little dilute *HCl* may be used to remove the last traces of the fused mass sticking to the inside of the crucible. The crucible should be washed out thoroughly, the fused mass and washings transferred to a casserole, considerable water added, and then *HCl* added—drop by drop, to avoid running over, as the carbonate effervesces violently with the acid. The addition of the acid is continued until the carbonate is all dissolved and the acid is in slight excess. Then, the casserole is covered with a watch glass and the solution evaporated down to dryness. The evaporation should be completed at a temperature not much above the boiling point of water; otherwise, the solution is apt to spit when it becomes thick, and thus lose some of the silica. After the mass is dry, the heat is raised somewhat, to drive off the last traces of *HCl*. More water is then added, as well as a few drops of *HCl*, and the whole is again boiled. This is then filtered and the residue washed several times with hot water, and then dried and ignited as in the case of the insoluble matter. The residue now consists entirely of silica, the weight of which is determined exactly as in the case of the insoluble matter.

For ordinary work, in which extreme accuracy is not sought, it will not be necessary to evaporate the solution down to dryness and again take up with water. The residue from the solution of the fused mass may be filtered off

directly, after boiling the solution, and then washed, ignited, and weighed as silica.

49. Assay of Silica Alone.—When silica alone is wanted, the following rapid method can be followed: One gram of the ore is dissolved with HCl, evaporated to dryness, and redissolved in dilute HCl. This is filtered on to an ashless filter, and the insoluble silicious residue washed, dried, ignited, and weighed. The material should be ignited in a platinum crucible, then a few drops of hydrofluoric acid, HF, and a few drops of concentrated H_2SO_4 added. This is then evaporated to dryness, when the silica will pass off as silicon fluoride. (The work should be done under a hood.) The residue is then ignited and weighed with the crucible. The difference between the two weights will be the weight of the silica direct as SiO_2.

If the insoluble silicious matter contains calcium, magnesium, potassium, or sodium, the loss in weight due to driving off the silica will not be entirely apparent, on account of the fact that some of the sulphuric acid will combine with the elements just named to form sulphates, and hence the amount of sulphuric acid so combined must be determined and added to the amount already found. This may be accomplished by fusing the residue with sodium carbonate and dissolving in water acidulated with a little HCl. This is heated to boiling and a hot solution of barium chloride, $BaCl_2$, added. The precipitated barium sulphate, $BaSO_4$, is then filtered off, ignited, and weighed. The amount of SO_3 in the barium sulphate can be calculated by multiplying the weight obtained by .3433. This should be added to the amount of SiO_2 already obtained. When the ore contains appreciable amounts of barium sulphate, this method is not admissible, on account of the fact that the SO_3 contained in the mineral barium sulphate would appear as SiO_2.

EXHAUSTIVE ANALYSIS OF IRON ORE

50. Treatment of the Ore.—To make an exhaustive analysis of an iron ore, it must be finely pulverized, and thoroughly dried at a temperature of 105° C. for 1 hour. A large sample (10 g. of the dried ore) is taken for analysis, in order to have sufficient material and solution to work with. The weighed ore is placed in a 6-inch porcelain dish covered with 50 c. c. of hydrochloric acid and 15 c. c. of nitric acid, and evaporated to dryness. After cooling the dish, 75 c. c. of hydrochloric acid is added, and, after warming, 100 c. c. of water is added and the mixture boiled and filtered into a one-half liter graduated flask. The residue is thoroughly washed on the filter with water and the contents brought up to the half-liter mark with the temperature at 15.5° C. There are now two portions of the original ore to be treated; namely, that in the solution and that which in the solid form was left on the filter.

51. Weight of Insoluble Matter.—The insoluble matter is dried to drive off all moisture and then ignited and weighed as gangue.

This is well illustrated by the following example:

	Grams
Insoluble residue and crucible weighed . .	10.551
Crucible alone weighed	10.301
Weight of insoluble residue	.250

Per cent. of insoluble residue $\frac{.250 \times 100}{10} = 2.5$ per cent.

If the insoluble matter contains substances besides silica, it should be fused with sodium carbonate in a platinum crucible. The residue should then be cooled, dissolved in water, acidified with hydrochloric acid, and evaporated to dryness in a 3-inch dish, when it is taken up with dilute hydrochloric acid, filtered, and washed well. The solid residue is probably silica, SiO_2, while the solution contains alumina, Al_2O_3. The silica is dried, ignited, and weighed as SiO_2. The alumina is thrown down by ammonia as aluminum hydrate, and is filtered, washed, ignited, and weighed as alumina, Al_2O_3.

52. Weight of Phosphorus Pentoxide, P_2O_5.—Two samples of 100 c. c. of the solution are evaporated to 30 c. c., cooled, made alkaline with ammonia, then slightly acidified with nitric acid, and poured with constant stirring into 40 c. c. of ammonium-molybdate solution, and allowed to stand 3 hours. This is then filtered and washed with water containing one-eighth its volume of the molybdate solution, the filtrate and washings being neglected. Ammonia to about 15 c. c. in quantity is poured on the yellow precipitate on the filter and the solution formed caught in a beaker. The filter paper free from yellow precipitate is washed with hot water, and the filtrate treated with hydrochloric acid, which precipitates yellow ammonium phosphomolybdate. Standard magnesia mixture to the amount of 30 c. c. is now added with constant stirring, and the beaker set aside for 30 minutes. This is filtered on ashless filter, washed with weak ammonia (1 to 8), dried, ignited in a porcelain crucible, and weighed as magnesium pyrophosphate. This operation is illustrated by the following:

	Grams	Grams
Crucible + $Mg_2P_2O_7$	7.6140	8.923
Crucible	7.6104	8.919
$Mg_2P_2O_7$	.0036	.004

$$Mg_2P_2O_7 : P_2O_5 = \frac{.0036 + .004}{2} : x$$

$$221.06 : 140.94 = .0038 : .0024$$

$$\text{The percentage of } P_2O_5 = \frac{.0024 \times 100 \times 5}{10} = .12 \text{ per cent.}$$

53. Weight of Alumina, Al_2O_3.—One hundred c. c. of the solution is diluted with 500 c. c. of water in an 800-c. c. flask and boiled, during which operation ammonium carbonate, $(NH_4)_2CO_3$, is slowly added until the solution becomes dark red in color, but the reaction must be faintly acid. About 25 c. c. of ammonium acetate, $NH_4C_2H_3O_2$, is added, and the solution is boiled and filtered. The residue contains alumina and iron oxide, while the solution contains manganese oxide, calcium oxide, and magnesia.

The residue is washed with water and dissolved in hydrochloric acid and given a bulk of 250 c. c. To 50 c. c. of this solution a slight excess of ammonia is added, then it is boiled, filtered, dried, ignited, and weighed as $Fe_2O_3 + Al_2O_3$. The amount of Fe_2O_3 in this solution is calculated. If the weight of the Fe_2O_3 is subtracted from the weight of the $Fe_2O_3 + Al_2O_3$, the difference will be Al_2O_3. As an illustration of this, the following may be taken:

The solution was 50 c. c. from 250 c. c. = one-fifth of 100 c. c.

	Grams
The crucible + Fe_2O_3 + Al_2O_3 weighed . .	12.6614
The crucible alone weighed	12.3160
$Al_2O_3 + Fe_2O_3$	.3454

By titration it was found that 50 c. c. of the original solution gave .8316 g. of Fe_2O_3, or .8316 ÷ 2.5 = .3326 g. of ferric oxide for 50 c. c. of the 250 c. c. solution of $Fe_2O_3 + Al_2O_3$.

.3454 − .3326 = .0128 g. of Al_2O_3 by difference. Then,

$$\frac{.0128 \times 100 \times 25}{10} = 3.2 \text{ per cent. of } Al_2O_3$$

54. Weight of Manganese Oxide.—The solution remaining from the precipitation of ferric oxide and alumina is placed in a flask and to it are added a few drops of bromine and ammonia. The mixture is shaken until all the bromine is dissolved, and then is set aside for 12 hours, after which it is filtered and washed with a 2-per-cent. solution of ammonium acetate. The residue is manganese dioxide, MnO_2. It is dried, ignited, and weighed as manganese oxide. Of the original solution, 100 c. c. was taken and this was diluted five times for the alumina determination. Assume that,

	Grams
The crucible and Mn_3O_4 weighed	12.166
The crucible alone weighed	12.131
Then, the Mn_3O_4 weighed, by difference,	.035

$$\frac{.035 \times 5 \times 100}{10} = 1.75 \text{ per cent. of } Mn_3O_4$$

55. Weight of Calcium Oxide.—To the solution left from filtering off MnO_2, ammonium carbonate and a few

drops of ammonia are added. This is set aside for 3 hours, then filtered, and washed. The residue on the filtrate is calcium acetate, CaC_2O_4. This is dried, ignited, and weighed as lime, CaO. Assume that:

	GRAMS
Weight of crucible + CaO	8.936
Weight of crucible alone	8.929
Difference	.007

Then, $\frac{.007 \times 5 \times 100}{10} = .35$ per cent. of CaO

56. Weight of Magnesia.—The solution from which the calcium acetate was precipitated is concentrated by heat to 250 c. c., and then an excess of sodium phosphate, Na_2HPO_4, is added with constant stirring, thus precipitating magnesium phosphate, $Mg_2P_2O_7$. This is allowed to stand for 2 hours, when it is filtered, dried, ignited, and weighed as magnesium pyrophosphate. Assume that:

	GRAMS
Crucible + $Mg_2P_2O_7$ weighed	8.929
Crucible alone weighed	8.919
$Mg_2P_2O_7$ weighs, by difference, ...	.010

Then, $Mg_2P_2O_7 : 2MgO = .01 : x$

or $221.06 : 80.12 = .01 : .00362$

As the original solution has been diluted five times,

$$\frac{.0036 \times 5 \times 100}{10} = .18 \text{ per cent. of } MgO$$

57. Weight of Sulphur Trioxide.—To ascertain the weight of sulphur trioxide, 50 c. c. of the original solution is boiled and an excess of barium chloride, $BaCl_2$, added. The solution is allowed to stand 3 hours and is then filtered through two ashless filters, when it is washed with hot water, dried, ignited, weighed as barium sulphate, $BaSO_4$, and calculated to SO_3. As an illustration, assume that:

	GRAMS
Crucible and barium sulphate weighed . .	11.126
Crucible alone weighed	11.011
Then, by difference, $BaSO_4$, weighs . . .	.115

$$BaSO_4 : SO_3 = .115 : x$$

or
$$231.74 : 79.46 = .115 : .0394$$

As one-tenth of the original solution was taken for this assay,

$$\frac{.0394 \times 10 \times 100}{10} = 3.94 \text{ per cent. of } SO_3$$

58. Weight of Water of Hydration.—Ores contain water in two forms: one form can be readily driven off by heat at a temperature slightly above 100° C., the boiling point of water; the other form is the water of crystallization or hydration. For example, 1.5 g. of ore is dried at 105° C., and transferred to a bulb tube, and connected with a Bennert drying apparatus procured at a chemical supply house. The **U** tube of the apparatus contains calcium chloride and before it is connected with the bulb tube it is carefully weighed. Heat is now applied to the bulb tube and increased gradually until the tube and the ore become red hot. Air travels slowly through the tube carrying the water of crystallization to the **U** tube, where it is absorbed by the calcium chloride. After heating at a red heat for 15 minutes, the **U** tube and its contents are weighed. The difference in weight is due to the absorption of the water of hydration. As an illustration:

	Grams
The amount of ore taken	1.5
Weight of $CaCl_2$ tube and water	29.065
Weight of $CaCl_2$ tube	28.963
Weight of water	.102

Then, $$\frac{.102 \times 100}{1.5} = 6.8 \text{ per cent. of water}$$

ELECTROLYTIC DETERMINATION OF COPPER

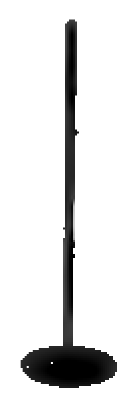
59. The electrolytic determination of copper is also sometimes called the *battery assay*. There are several slightly different methods of conducting the electrical determination of copper, but the following is a good general method, especially where zinc and antimony are not present in large quantities: The sample should be passed through

an 80-mesh sieve. When determining the contents of copper mattes, $\frac{1}{2}$ g. is sufficient, while in the case of the ores the amount taken may vary from 1 to 2 g., depending on the richness of the ore, and in the case of slags 3 g. may be taken. After weighing the samples, they are placed in beakers and slightly moistened with cold distilled water. Then 25 c. c. of strong nitric acid and 10 to 15 drops of strong sulphuric acid are added. The beakers should be covered with watch glasses and set on a sand bath, where they are heated until the nitrous fumes have all passed off and the copper is in solution. The watch glass is washed down into the beaker and the solution evaporated until choking white fumes of sulphuric acid appear. The beakers are then set to one side to cool, after which the mass is moistened with dilute nitric acid (sp. gr. 1.20), about 6 or 7 c. c. being used. Four or 5 drops of sulphuric acid and 40 c. c. of water are also added. The whole is then heated on the sand bath until the mass is in solution, after which the insoluble residue is filtered off. The residue should be examined to see that there are no copper minerals left undissolved. The filtrate is saved in the beaker ready for precipitating by means of electricity.

The electrical energy necessary to electrolyze the copper solution may be furnished by one of the various forms of batteries or from a dynamo. The gravity cell is probably the most common form of battery employed for this purpose, but a Grove cell will usually give better results, even though it requires more care in its manipulation. It is best to have a surplus of electrical energy, but too strong a current must be guarded against. Ordinarily, three Grove cells, freshly made up, will furnish a current sufficient to deposit the copper from six to eight solutions, none of which contains more than .5 g. of copper in 1 g. of the sample.

60. The copper is deposited on a platinum plate called a *cathode*, and another platinum surface is provided to act as the other pole for the electricity, and this is called an *anode*. Fig. 1 (*a*) illustrates a very convenient form of cathode,

which is composed of a plain platinum cylinder about 2½ inches long and 1 inch in diameter. The rod that supports it is 4½ inches long and usually a little less than $\frac{1}{16}$ inch in diameter. Such a cathode would weigh from 16 to 18 g. It is well to have the cathodes as large as this, on account of the fact that they afford ample surface for the precipitation of the copper without its accumulating in a spongy mass, which would cause loss while weighing. The anode may be made from platinum wire, as shown in (*b*). The wire is a little less than $\frac{1}{16}$ inch in diameter, and the straight part is about 7 inches in length. The lower end of the wire is twisted into a spiral, which is situated below the cathode in the solution to be treated. The diameter of the spiral coil is about 1 inch. One advantage of this form of anode is that it affords a uniform evolution of gas throughout the solution, and hence tends to precipitate copper on both the inside and the outside of the cathode in an even manner.



(*a*) (*b*) (*c*)

Fig. 1

The solution to be treated is placed in a deep beaker similar to that shown in (*c*). The cathode is connected with the positive pole of the battery, and the anode with the negative or zinc pole of the battery. The cathode should not be completely immersed in the solution to be treated, and when it is supposed that all the copper has been deposited, the cathode can be immersed deeper in the solution and the current allowed to run for half an hour longer. If any copper deposits on the clean surface, it will show at once that all the metal has not been removed from the solution. After all the copper has been deposited, the anode is loosened and the beaker and anode removed, leaving the cathode with the deposit of copper hanging to the wire connected with the battery. The cathode is next washed with distilled water and then immersed in alcohol. What alcohol adheres to it is burned off so as to ignite and dry the cathode. The

copper should appear of a rosy or pinkish color. After the cathode has cooled to the temperature of the room, it is weighed, and the difference in weight between the bare cathode and this second weighing is taken as the amount of copper in the sample treated.

Too strong a current will cause too strong an evolution of gas, and the copper will deposit dark-colored, while if zinc is present it may deposit on the copper. By being careful with the work and determining the conditions best suited to the products being handled, it is possible to do extremely accurate work by means of the battery assay. Copper may be completely deposited and removed from solutions containing iron, aluminum, manganese, zinc, nickel, cobalt, chromium, cadmium, lime, barium, strontium, and magnesium. The solution may subsequently be employed for the determination of other elements that it contains. The cathode may be cleaned by simply dissolving the deposited copper in nitric acid. After all the copper is dissolved, the cathode should be carefully washed to remove any acid or copper salts before it is again employed for a determination.

SLAG ANALYSIS

61. Furnace slags are analyzed to ascertain whether the fluxes necessary to a fluid slag are in the right proportion; also, to ascertain the quantity of metal that is being lost in the slag. In many cases, a complete analysis is made of the slag, particularly in lead and copper smelting where the stock is liable to vary in composition. In most cases, only a partial analysis of slag is required. In iron furnace slags, the constituents most frequently determined are the silicates, phosphorus, and sulphur, the ferrous oxide going into slag. In lead and copper smelting, the number of oxides to be determined are silica, ferrous oxide, lime, magnesia, alumina, and sometimes zinc, manganese, barium, lead, or copper.

It will not be necessary to go into the details of the operations connected with slag analyses, and therefore the scheme to be followed is merely outlined.

62. Iron Furnace Slags.—While some slags are soluble in acids, it is better to proceed, so far as is consistent with economical work, to make a gravimetric analysis.

Two g. of the finely powdered slag is fused with 10 g. of sodium carbonate and 1 g. of sodium nitrate. The fusion is to be made in a platinum crucible over a Bunsen blast lamp, and should take about one-quarter hour. The crucible and its contents after cooling are transferred to a No. 4 beaker containing 100 c. c. of warm water. The water will permit the contents of the crucible to be extracted, and then the crucible is removed. The liquid in the beaker is acidified with hydrochloric acid, and the contents carefully transferred to a 6-inch porcelain crucible and evaporated to dryness. Then, 50 c. c. of hydrochloric acid and 100 c. c. of water are added, and the whole boiled and filtered into a 250-c. c. graduated flask. It is next washed well with hot water, cooled to 15.5° C., and water added to the 250 c. c. mark, and the whole thoroughly mixed.

The plan to pursue is about as follows:

1. Dry, ignite, and weigh the residue as silica.
2. Take 50 c. c. of the solution and treat for phosphorus.
3. Take 50 c. c. of the solution and titrate for iron.
4. Take 50 c. c. of the solution and treat for alumina.
5. Take 50 c. c. of the solution and treat for MnO_2 and treat the filtrate for CaO by the ammonium-oxalate method, then filter, and treat the resulting filtrate for MgO.
6. Take 50 c. c. of the solution and with $BaCl_2$ precipitate the sulphur as barium sulphate.

All of these operations have been explained.

63. Lead and Copper Furnace Slags.—The constituents of lead and copper slags are about the same, the analysis only differing in looking for lead in lead furnace slag and for copper in copper furnace slag. The plan to pursue is as follows:

1. Determine the silica by gravimetric analysis.
2. Determine the barium as follows: Treat .5 g. of slag with dilute hydrochloric acid and a few drops of nitric and

sulphuric acids. Evaporate to dryness, add water and hydrochloric acid, filter, wash, dry, ignite, and weigh. The difference between this weight and the weight of silica in 1 will represent the weight of the barium sulphate.

3. Determine the lime in the filtrate from 1 and 2.

4. Determine the iron from .5 g. of slag by the volumetric method.

5. Determine the manganese in 1 g. of slag by Volhard's method.

6. Determine the zinc volumetrically from 1 g. of slag.

7. Determine the lead volumetrically, using 5 g. of slag.

8. Determine the silver in 1 A. T. of slag by the crucible fire-assay.

Instead of making so many different operations, the chemist can use 10 g. of finely powdered slag, fuse it with sodium carbonate and a little niter, and proceed as in the determination of iron furnace slags. The barium and silver, however, will have to be analyzed as just described.

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NOTE.—All items in this index refer first to the section, and then to the page of the section. Thus, "Abutments, §46, p49," means that abutments will be found on page 49 of section 46.

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